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THE ZINC INDUSTRY IN AMERICA

By J. A. Van Mater

Annual Meeting, Toronto, 1915

It is not known to whom the isolation of zinc is due but it was mentioned by Paracelsus (1493-1541). In 1721 Henckle published his discovery of the fact that it could be obtained from calamine, and he is named by Beckman as the first to *intentionally* carry out the process. It seems quite certain that zinc was made in the East at a much earlier date, but authorities differ as to whether China or India was the birthplace of the process. The earliest attempt seems to have been the production of brass by the cementation of calamine ore (silicate of zinc), with native copper. Later metallic zinc, or spelter, was made, and about 1740 the art was introduced into England. In 1798 it was introduced into Silesia, Germany, by Ruhberg, (who went to England to learn about it), and into Carinthia, Austria, about the same time by Dillinger.

The production of spelter in Belgium began with the experiments of Abbé Daniel Dony in 1780. Dony was an experienced chemist, whose success was finally achieved accidentally after a long series of disappointing results. While endeavouring to extract zinc from calamine by fusion, in a reverberatory furnace, it occurred to him that the heat applied was not sufficiently high and he added coal dust to the mineral to attain a higher temperature. In order to observe what took place in the interior of his furnace, he built an ordinary flower-pot into one side to serve as a peep-hole through which he could watch the operation. The furnace having been charged and the firing begun, drops of zinc were observed to condense on the inside of the flower pot, which was cooler than the furnace itself. The method of extracting the metal by distillation was thus discovered. Dony obtained from the Imperial Government a grant of the Vieille Montagne mine, with the obligation of making experiments for the production of spelter, and about 1805 built

furnaces at Liège and commenced the production of zinc on a commercial scale. This was the beginning of what is today the largest zinc company in the world.

The first attempt to smelt zinc ore in America, of which we have any record, was in 1830, when some of the zincite, or red oxide of zinc ore, found at Franklin Furnace, New Jersey, was carefully selected and sent to the Arsenal at Washington, together with some selected ore from Perkiomen, Pennsylvania, where it was reduced in admixture with copper to make brass from which the standard weights and measures of the Government, ordered by Congress, were manufactured.

In 1845 Theodore William Roepper, a local mineralogist, who later became the first Professor of Mining and Geology in Lehigh University, while taking a stroll one afternoon in the vicinity of the Friedensville mine near Bethlehem, Pennsylvania, picked up a few specimens of the hitherto unknown mineral which he determined to be calamine. He thereupon conducted some experiments in Lehman's foundry at South Bethlehem, where he succeeded in making brass from the calamine and native copper. He, however, did not succeed in making spelter.

In the same year Andrew Wittman, after studying Overman's Metallurgy, conducted some experiments with ore from the Friedensville mine by means of a small crucible in a stove and succeeded in obtaining a few globules of metal, but evidently did not know that it was spelter.

In 1846 Roepper induced Robert Earp, a Philadelphia importer, to examine the deposit at Friedensville, after which Earp obtained a lease on the Ueberroth farm. He mined nine tons of ore and shipped it to England in one of his vessels. An attempt was made to reduce this ore at one of the English works but they were not successful in producing spelter, probably because they had become accustomed to smelting roasted ores requiring a lower temperature for reduction than the calamine.

Between 1854 and 1859 Colonel Samuel Wetherill made attempts to produce spelter from this same ore. His method was to heat the ore in an open hearth and draw the vaporized oxide through incandescent anthracite to reduce the oxide to

metal. He made a few tons in this way but the process was not economical and the experiments were discontinued.

In 1857 Matthiessen and Hegeler, two young graduates from the Freiburg School of Mines in Saxony, obtained permission to experiment in the oxide plant which had been erected at Friedensville. They succeeded in making spelter, but as they were not able to negotiate satisfactory terms with the owners of the property for the erection of a commercial plant, they abandoned the project.

In 1859, Joseph Wharton contracted with the Lehigh Zinc Company, the owners of the Friedensville mine, for the erection of spelter works of the Belgian type and brought to this country, Louis de Gée, of Ougrèe, Province of Liège, Belgium, to superintend their construction. The Belgian furnace proved to be successful and in July, 1859, the first spelter was produced. So far as I am aware this was the first spelter to be produced commercially in this country.

Mathiessen and Hegeler, after failing to make satisfactory arrangements with the Lehigh Zinc Company, went to La Salle, Ill., and erected works in 1859, which have been in continuous operation ever since, with the exception of slight suspensions between 1860 and 1862. They were the originators of the large type of furnace now so generally used in the West.

A few years later the Illinois Zinc Company built works at Peru, just west of La Salle, Illinois. These were, next to those at La Salle, the largest in the country. Later, at both of these works, rolling mills were erected and for a number of years they have converted about 50 per cent of their spelter into sheets. They are the only sheet zinc works in the country at the present time, with the exception of the small mill of Ball Brothers at Muncie, Indiana, where they roll zinc strips for fruit jar tops.

The rolling mill at La Harpe, Kansas, was built in 1902 by the Lanyon Zinc Company. It was a typical tin plate mill not well adapted to the rolling of sheet zinc and operated only a short time. This was later purchased by Ball Brothers and moved to Muncie.

The Lehigh Zinc Company at Bethlehem, Pa., built a zinc rolling mill in 1865, operating until 1877. The Passaic Zinc Company, built a rolling mill at their plant in Jersey City, New Jersey, about 1878. This ran intermittently for a year or two and was finally abandoned.

About 1865 the New Jersey Zinc Company erected spelter furnaces at their oxide plant in Newark, New Jersey, where they made a high grade spelter from refuse oxide. This was the beginning of Horse Head spelter, noted throughout the world for its purity, fluidity, and softness. Later this metal was made directly from willemite, one of the constituent minerals of the ore mined at Franklin, New Jersey.

Between 1864 and 1870, nine or ten zinc smelters were established, but I am unable to give the respective dates. Among these were the zinc works in the neighbourhood of St. Louis, the Missouri Zinc Works and the Carondelet Zinc Works in Missouri, and the Collinsville Works on the eastern side of the river in Illinois. They used Illinois coal and drew their ore supply from Southeastern and Southwestern Missouri.

Later a group of works was built on the bituminous coal belt extending from Indian Territory, through Eastern Kansas, into Northern Missouri and south along lines of the railroad penetrating into the lead region in Southwest Missouri. The oldest of these, begun in 1873, was at Weir City, Kansas.

R. and S. H. Lanyon started two furnaces at Pittsburgh, Kansas, in 1878. In 1880 this partnership was dissolved and S. H. Lanyon started the construction of a new works. In 1882 W. and J. Lanyon put up two similar furnaces in the immediate neighbourhood.

The same year the Granby Mining and Smelting Company began the erection of a works, using Siemen's gas furnaces, and the West Joplin Lead and Zinc Company built works at Joplin, Missouri. The Southwestern Lead and Zinc Company also erected works at Rich Hill, Missouri, about eighty miles from Joplin.

The Bertha Zinc Mines were first opened in 1879, when a small shipment of the ore was sent to Providence, Rhode Island, and there smelted. The resulting metal proved to be of such

rare purity that attention was at once attracted to this ore and a small smelting plant was built at Martin's Tank, Virginia, now Pulaski, in 1880. The ore was at first hauled in waggons twenty miles over the mountain to the works. In 1886 the works were remodeled and enlarged, consisting of ten large Welsh-Belgian furnaces of one hundred and forty retorts each, with an annual capacity of 3700 tons. The works were abandoned in 1911.

The discovery of semi-anthracite coal near Pulaski, which was found to be suitable both for mixing with the charge as reducing agent and as firing coal, doubtless influenced the location of these works, but upon the development of the famous Pocahontas coalfield, this coal was substituted for firing purposes.

The Bertha grade of metal made from this ore was similar in character to that made by the New Jersey Zinc Company from the Franklin ore. It, likewise, had a world-wide reputation and was in demand for the manufacture of high grade brass and bronze; also for fine spelter castings. The metal was free from arsenic, antimony and cadmium and contained only a trace of lead, being 99.96 to 99.98% zinc.

In 1892 spelter works were erected at Ingalls and Marion, Indiana, utilizing the natural gas for fuel, but as the supply decreased the price of gas became too high for smelting purposes. These works were closed down about 1904.

Shortly after the discovery of natural gas in Kansas, W. and J. Lanyon erected the first works in that field at Iola in 1896. Between 1896 and 1902 some twelve additional smelting plants were erected at Gas, Cherryvale, Neodesha, La Harpe, Chanute and Altoona.

All of these works were cheaply constructed and were of rather a temporary character, as it was uncertain just how long the natural gas would be available. The decrease in the supply of gas in this district caused the zinc smelting industry to seek new gasfields further southwest about 1906.

In 1905 the Bartlesville Zinc Company erected works at Bartlesville, Oklahoma. This was quickly followed by the erection of works by the Lanyon-Starr Smelting Company and

the National Zinc Company at the same point. During the next two years works were erected at Caney and Dearing, Kansas.

In 1908 the Tulsa Fuel and Manufacturing Company erected a plant at Collinsville, Oklahoma. The Bartlesville Zinc Company followed with a similar plant a few months later at Collinsville.

The United States Zinc Company built a plant at Blende, a suburb of Pueblo, Colorado, in 1903, to utilize low grade ores of the Rocky Mountain region carrying precious metal values. This plant was not at first a financial success, but later established a satisfactory practice and has since run continuously.

In 1904 the Grasselli Chemical Company built a ten-block plant at Clarksburg, West Virginia, using natural gas for fuel. This was followed by the Clarksburg Zinc Company's plant shortly after. This company does not smelt ore—they treat only dross and galvanizer's residues.

The Mineral Point Zinc Company constructed a three-block plant at Depue, Illinois, in 1906, which was afterward increased to six blocks and later to twelve.

The United Zinc and Chemical Company built at Springfield, Illinois, in 1907. The same year Hegeler Brothers built at Danville, Illinois, the plant being put in operation in 1908.

The Grasselli Chemical Company erected a plant at Meadowbrook, West Virginia, in 1911.

The American Zinc, Lead and Smelting Company completed a plant at Hillsboro, Illinois, and put same in operation in 1913. In 1914 the Tulsa Spelter Company completed and put in operation a plant at Sand Springs, Oklahoma.

The American Zinc and Chemical Company started up their new plant at Langeloth, Pennsylvania, later in 1914. The Granby Mining and Smelting Company's plant at East St. Louis, Illinois, although completed, was not put in operation until January, 1915, due to the low metal market.

In 1898 the New Jersey Zinc Company built a four-furnace plant at Palmerton, Pennsylvania, for the manufacture of high

grade metal from willemite. This was increased from time to time until in 1912 there were twenty-four furnaces.

ZINC SMELTING CAPACITY OF THE UNITED STATES

(Number of Retorts at End of Years)

Name	Situation	1913	1914
Altoona Zinc Smelting Co.	Altoona, Kan.	3,300	[2]3,840
American Zinc & Chem. Co.	Langeloth, Pa.	.....	864
American Zinc Co. of Ill.	Hillsboro, Ill.	3,200	4,000
American Zinc, Lead & Smg. Co.	Dearing, Kan.	3,840	[1]3,840
American Zinc, Lead & Smg. Co.	Caney, Kan.	3,648	[2]3,648
Bartlesville Zinc Co.	Bartlesville, Okla.	5,184	5,184
Bartlesville Zinc Co.	Collinsville, Okla.	8,064	8,064
Chanute Zinc Co.	Chanute, Kan.	[2]1,280	[1]1,280
Clarksburg Zinc Co.	Clarksburg, W. Va.	2,712	[4]2,712
Collinsville Zinc Co.	Collinsville, Ill.	[2]1,536	[1]1,536
Edgar Zinc Co.	St. Louis, Mo.	2,000	1,100
Edgar Zinc Co.	Cherryvale, Kan.	4,800	4,800
Granby Mining & Smelting Co.	Neodesha, Kan.	3,760	3,840
Granby Mining & Smelting Co.	E. St. Louis, Ill.	.....	3,240
Grasselli Chemical Co.	Clarksburg, W. Va.	12,672	8,772
Grasselli Chemical Co.	Meadowbrook, W.Va.		
Hegeler Zinc Co.	Danville, Ill.	1,800	[4]1,800
Illinois Zinc Co.	Peru, Ill.	4,640	[4]4,640
La Harpe Spelter Co.	La Harpe, Kan.	1,856	[4]1,856
Robt. Lanyon Zinc & Acid Co.	Hillsboro, Ill.	1,600	1,840
Lanyon-Starr Sm. Co.	Bartlesville, Okla.	3,456	3,456
Matthiessen & Hegeler Zinc Co.	LaSalle, Ill.	5,256	5,256
Mineral Point Zinc Co.	Depue, Ill.	6,800	9,080
National Zinc Co.	Bartlesville, Okla.	4,480	4,260
National Zinc Co.	Springfield, Ill.	3,200	3,200
Nevada Zinc Co.	Nevada, Mo.	[1]648	[3]
New Jersey Zinc Co.	Palmerton, Pa.	5,760	5,760
Pittsburgh Zinc Co.	Pittsburg, Kan.	[2]910	[1]910
Prime Western Spelter Co.	Gas City, Kan.	4,768	4,768
Sandoval Zinc Co.	Sandoval, Ill.	896	896
Tulsa Fuel & Mfg. Co.	Collinsville, Okla.	6,232	6,232
Tulsa Spelter Co.	Sand Springs, Okla.	.....	4,000
United States Zinc Co.	Pueblo, Colo.	1,920	1,920
Totals	.....	110,218	120,494

[1]Inactive throughout year.
[2]Inactive during latter part of year.
[3]Plant being dismantled.
[4]No report received; entered the same as last year.

PRODUCTION

The earliest reliable statistics of the spelter production in this country began with the year 1873. The production in that year amounted to 7,343 tons. In the next two years the production doubled, amounting, in 1875, to 15,833 tons.

The following tables showing production were taken from the United States Geological Survey reports:

ANNUAL PRODUCTION OF SPELTER IN THE UNITED STATES FROM DOMESTIC AND FOREIGN ORES, 1873-1913, IN SHORT TONS.

Year	From Foreign Ore	From Domestic Ore	Total	Year	From Foreign Ore	From Domestic Ore	Total
1873	.....	7,343	7,343	1897	.....	99,980	99,980
1875	.....	15,833	15,833	1898	.....	115,399	115,399
1880	.....	23,239	23,239	1899	.....	129,051	129,051
1882	.....	33,765	33,765	1900	.....	123,886	123,886
1883	.....	36,872	36,872	1901	.....	140,822	140,822
1884	.....	38,544	38,544	1902	.....	156,927	156,927
1885	.....	40,688	40,688	1903	.....	159,219	159,219
1886	.....	42,641	42,641	1904	.....	[1]186,702	186,702
1887	.....	50,340	50,340	1905	.....	[1]203,849	203,849
1888	.....	55,903	55,903	1906	25,076	199,694	224,770
1889	.....	58,860	58,860	1907	26,115	223,745	249,860
1890	.....	63,683	63,683	1908	19,675	190,749	210,424
1891	.....	80,873	80,873	1909	25,535	230,225	255,760
1892	.....	87,260	87,260	1910	16,705	252,479	269,184
1893	.....	78,832	78,832	1911	14,905	271,621	286,526
1894	.....	75,328	75,328	1912	14,899	323,907	338,806
1895	.....	89,686	89,686	1913	9,424	337,252	346,676
1896	.....	81,499	81,499				

[1]Including an unknown quantity smelted from foreign ore, small in 1904, but larger in 1905.

Production of primary spelter in the United States in 1906-1913, apportioned according to source of ore, in short tons.

SOURCE	QUANTITY								Percentage of total in 1913
	1906	1907	1908	1909	1910	1911	1912	1913	
Arizona	64	77	152	2,862	2,594	2,476	4,092	4,675	1.38
Arkansas	1,801	1,911	2,002	135	286	385	604	478	.14
California	.....	140	27	.....	.....	959	1,672	1,012	.30
Colorado	32,456	26,077	24,885	20,121	23,238	42,233	60,841	58,113	19.23
Idaho	573	3,508	581	901	3,454	3,180	6,800	10,190	3.02
Illinois	282	1,446	298	675	1,551	2,884	3,952	1,345	.40
Iowa	201	220	516	.....	40	.....	.....	.....	....
Kansas	3,902	13,850	8,628	9,185	10,220	6,843	5,668	9,956	2.95
Kentucky	335	358	341	22	29	205	394	172	.05
Maine	.....	1	.....	.....	.....	.....	.....	.....	....
Missouri	130,348	141,824	123,655	140,676	140,652	127,540	149,557	129,018	38.26
Montana	1,415	.....	900	4,725	12,408	22,115	14,196	35,604	10.56
Nevada	1,768	1,692	398	825	1,516	1,595	6,132	5,828	1.73
New Jersey	11,206	13,573	6,926	16,035	20,217	15,128	16,941	24,247	7.19
New Mexico	555	136	134	3,961	4,911	3,778	6,882	3,765	1.12
North Carolina	.....	.....	.....	.....	.....	.....	.....	152	.04
Oklahoma	.....	719	2,235	3,008	2,297	2,247	2,041	6,397	1.90
Tennessee	124	181	341	695	925	1,000	1,935	2,635	.78
Texas	8	16	.....	.....	.....	.....	245	303	.09
Utah	2,449	1,972	282	5,960	7,221	7,004	7,756	9,503	2.82
Virginia	1,143	771	910	58	1,168	228	62	116	.03
Washington	7	.....	.....	.....	.....	12	.....	.....	....
Wisconsin	11,057	15,273	17,538	20,381	19,752	31,809	34,137	33,743	10.01
Total from domestic ores	199,694	223,745	190,749	230,225	252,479	271,621	323,907	337,252	100.00
Foreign—									
Canada	201	545	2,425	978	3,304	1,598	4,199	1,424	....
Mexico	24,875	25,570	17,250	24,557	13,401	13,307	10,700	6,205	....
Europe	.....	.....	.....	.....	.....	.....	.....	1,175	....
Siberia	.....	.....	.....	.....	.....	.....	.....	620	....
Total from foreign ores	25,076	26,115	19,675	25,535	16,705	14,905	14,899	9,424	....
Grand Total	224,770	249,860	210,424	255,760	269,184	286,526	338,806	346,676	....

Production of spelter in the United States, 1873-1913, apportioned according to locality in which smelted, in short tons.

Year	Illinois	Kansas	Missouri	Oklahoma	Eastern	Southern	Colorado	Total
1873	.....	.....	....	....	....	....	....	7,343
1875	6,678	600	4,650	....	3,305	....	....	[1]15,833
1880 (census)	10,904	2,451	5,671	....	4,213		....	23,239
1882	18,201	7,366	2,500	....	5,698		....	33,765
1883	16,792	9,010	5,730	....	5,340		....	36,872
1884	17,594	7,859	5,230	....	7,861		....	38,544
1885	19,427	8,502	4,677	....	8,082		....	40,688
1886	21,077	8,932	5,870	....	6,762		....	42,641
1887	22,279	11,955	8,660	....	7,446		....	50,340
1888	22,445	10,432	13,465	....	9,561		....	55,903
1889	23,860	13,658	11,077	....	10,265		....	58,860
1890	26,243	15,199	13,127	....	9,114		....	63,683
1891	28,711	22,747	16,253	....	8,945	4,217	....	80,873
1892	[2]31,383	24,715	16,667	....	9,582	4,913	....	87,260
1893	[2]29,596	22,815	13,737	....	8,802	3,882	....	78,832
1894	[2]28,972	25,588	11,992	....	7,400	1,376	....	75,328
1895	[2]35,732	25,775	14,998	....	9,484	3,697	....	89,686
1896	[2]36,173	20,759	14,001	....	8,139	2,427	....	81,499
1897	[2]37,876	33,396	18,125	....	7,218	3,365	....	99,980
1898	[2]47,103	40,132	19,533	....	8,631		....	115,399
1899	[2]50,118	52,021	18,107	....	8,805		....	129,051
1900	[2]38,750	62,136	14,741	....	8,259		....	123,886
1901	[2]44,896	74,240	13,083	....	8,603		....	[1]140,822
1902	[2]47,096	86,564	11,087	....	12,180		....	[1]156,927
1903	[2]47,659	88,388	9,994	....	12,301		877	[1]159,219
1904	47,740	107,048	12,150	....	14,893		4,871	[1]186,702
1905	46,606	114,287	11,844	....	24,513		6,599	[1]203,849
1906	47,939	129,564	11,077	....	29,930		6,260	224,770
1907	56,056	134,108	11,732	5,035	37,626		5,303	249,860
1908	50,244	99,298	10,201	14,864	32,738		3,079	210,424
1909	67,653	103,299	7,037	28,782	42,941		6,048	255,760
1910	73,038	105,697	5,571	34,760	43,606		6,512	269,184
1911	83,130	98,413	4,116	46,315	47,155		7,397	286,526
1912	88,397	101,104	7,085	76,925	56,252		9,043	338,806
1913	106,654	74,106	5,085	83,214	69,063		8,554	346,676

[1]Secondary spelter included; 600 tons in 1875; 2,716 tons in 1901; 2,675 tons in 1902; 3,302 tons in 1903; 3,300 tons in 1904; and 1,469 tons in 1905.
[2]Includes also the production of smelters in Indiana.

Prior to 1907 we have no reliable statistics of the secondary spelter produced from galvanized dross and skimmings, etc., a portion of which was resmelted and a portion redistilled. Since then it has been as follows:

1907.....18841 Short Tons
1908.....16970 " "

1909	33040	Short Tons.
1910	41223	" "
1911	40513	" "
1912	52251	" "
1913	50005	" "

From the first statement covering the production in the United States, it will be noted that there was a steady increase in production until 1893 and 1894, when there was a decline. 1896 likewise showed a decline, as did 1900, 1906 and 1908. The increase from 1873 to 1883 was 402% from 1883 to 1893, 114%; from 1893 to 1903, 102% 1903 to 1913, 124%.

From the second table it will be noted that during the past eight years over 38% of the metal made from domestic ores came from Missouri. The ore production from that state was about the same in 1913 as in 1906, with slight fluctuations in the interval. In 1906 over 32,000 tons of spelter came from Colorado ore, while in 1913 this had increased to over 58,000 tons. Wisconsin has increased from 11,057 tons in 1906 to 33,743 tons in 1913. Spelter made from imported Canadian ores in 1906 is given as 201 tons; in 1913 it had risen to 1,424 tons. Spelter made from imported Mexican ore in 1906 amounted to 24,875 tons and fell in 1913 to 6,205 tons. This falling-off in the importation of ore from Mexico has been due to two causes: First, the high protective tariff of 1909 in the United States; second, the civil conflict in Mexico.

From the third statement it will be noted that from the beginning Illinois led all the other states in the production of spelter, with Kansas a close second, until 1899, when Illinois produced 50,118 tons and Kansas, 52,021. Missouri had passed its zenith the year before with 19,533 tons. Kansas then held the lead until 1912, when she produced 101,104 tons. In the meantime Illinois had increased to 88,397 tons and in the next year had risen to 106,654 tons, Kansas dropping back to 74,106 tons. Oklahoma entered the producing list in 1907 with 5,035 tons and increased to 83,214 tons in 1913.

The estimated production of primary spelter for 1914 is 360,000 tons, the largest in the history of the industry, in spite of the fact that several plants were idle throughout the year and

a number of plants operating at only partial capacity, due to the low price of metal.

The zinc smelting capacity in this country, as well as the production of spelter, has of late years increased faster than the consumption, as the following table will show, and had it not been for the great demand from abroad, caused by the war, the spelter market in this country would now be very flat.

Zinc-smelting capacity, production and apparent consumption of spelter in the United States, 1908-1914.

Year	Retorts at Close of Year		Production	Apparent Consumption	Surplus Production
	Number	Estimated Maximum Annual Capacity			
		Short Tons	Short Tons	Short Tons	Short Tons
1908	93,958	404,960	210,424	214,167	3,743
1909	94,328	400,681	255,760	270,730	14,106
1910	93,722	398,105	269,184	245,884	36,466
1911	85,018	361,327	286,526	280,059	30,308
1912	107,948	458,779	338,806	340,341	18,661
1913	111,458	473,697	346,676	295,370	61,928
1914	127,946	543,771	[1]360,000	[1]285,000	75,000

[1]Estimated.

In the foregoing table the estimated maximum annual capacity is based upon the use of high grade ores, but this will be reduced because many of the plants treat low grade ores. Likewise, a certain portion of the capacity is taken up in the redistillation of the zinc dross, so that the spelter produced from ore always falls considerably short of the maximum capacity. The production of ore has correspondingly increased.

The table following shows the high, the low and the average New York spelter market for the past thirty-one years. To obtain the average St. Louis spelter market for the same period deduct 15 cents in each instance. This deduction represents the difference in freight. The freight on spelter from St. Louis to New York was increased December 16, 1914, to 17 cents per cwt.

SPELTER PRICES IN NEW YORK

Highest, lowest and average yearly prices of spelter in New York.

YEAR	Highest	Lowest	Average
1884	4.62½	4.00	4.44
1885	4.62½	4.00	4.34
1886	4.60	4.25	4.40
1887	5.62½	4.40	4.62½
1888	5.50	4.50	4.91
1889	5.40	4.62½	5.02
1890	6.17½	5.00	5.55
1891	6.00	4.65	5.02
1892	4.90	4.35	4.63
1893	4.50	3.55	4.08
1894	4.00	3.25	3.52
1895	4.35	3.10	3.63
1896	4.25	3.60	3.94
1897	4.35	3.75	4.12
1898	5.30	3.87½	4.57
1899	7.00	4.45	5.75
1900	4.80	4.10	4.40
1901	4.50	3.90	4.08½
1902	5.62½	4.10	4.90
1903	6.25	4.70	5.62
1904	6.12½	4.75	5.17
1905	6.65	5.30	6.00
1906	6.85	6.00	6.27½
1907	7.25	4.35	6.20½
1908	5.17½	4.35	4.74
1909	6.50	4.75	5.52
1910	6.35	5.25	5.66
1911	7.00	5.50	5.91
1912	7.65	6.45	7.11
1913	7.35	5.10	5.80
1914	5.59	4.91	5.21
Average for 31 years	.....	.....	5.007

The average price for spelter, New York, for the ten-year period, 1885 to 1894, inclusive, was $4.60; 1895 to 1904, inclusive, $4.61; 1905 to 1914, inclusive, $5.84.

The American Society for Testing Materials adopted in June, 1911, the following standard specifications for primary, or virgin, spelter:

A. HIGH GRADE.—The spelter shall not contain over

0.07 per cent lead
0.03 " " iron
0.05 " " cadmium.

It shall be free from aluminum.

The sum of the lead, iron and cadmium shall not exceed 0.10 per cent.

B. INTERMEDIATE.—The spelter shall not contain over

0.20 per cent lead
0.03 " " iron
0.50 " " cadmium.

It shall be free from aluminum.

The sum of the lead, iron and cadmium shall not exceed 0.50 per cent.

C. BRASS SPECIAL.—The spelter shall not contain over

0.75 per cent lead
0.04 " " iron
0.75 " " cadmium.

It shall be free from aluminum.

The sum of the lead, iron and cadmium shall not exceed 1.20 per cent.

D. PRIME WESTERN.—The spelter shall not contain over

1.50 per cent lead
0.08 " " iron.

ZINC ORE SOURCES

Based upon the metallic content of the ore, Missouri still continues to be by far the largest source of ore supply for the manufacture of spelter, with Colorado second. Montana now stands third and will doubtless, within the present year, become the second largest producer. Wisconsin occupies fourth place, with New Jersey fifth.

It should be noted, however, that only about 14% of the ore from the mines at Franklin, New Jersey, is used for manufacturing spelter, the balance of the output being used for the manufacture of zinc oxide.

Of the spelter made from domestic ores the above five States produce about 90 per cent. Considerable ore was imported into

the United States in the few years prior to the summer of 1909, when the tariff act of that year, imposing a heavy duty on zinc-bearing ore, very largely restricted importations.

Under that act all zinc-bearing ore containing less than 10% Zn. was admitted free; over 10% to 20%, there was a duty of $\frac{1}{4}$ cent per lb. on the zinc contained therein; 20% to 25% $\frac{1}{2}$ cent per lb. on the zinc contained therein; over 25% Zn. 1 cent per lb. on the zinc contained therein There was a further duty on any lead content, even though this is an objectionable element. Under normal markets this duty was practically prohibitive.

The Tariff Act of 1913 substituted *ad valorem* for specific duties and provided that on all importations of zinc-bearing ores the duty be estimated at the port of entry. The duty was fixed at 10 per cent *ad valorem* upon the zinc contained therein.

With the more favourable tariff importations of ore from Canada have materially increased during the past year, but on account of the civil conditions in Mexico there has been almost a suspension of shipments from that country.

Metallurgy of Zinc

Improvement in zinc metallurgy is one of operation rather than radical change, such as better retorts, closer attention to reduction fuel with regard to the ash ingredients, better blending of the ores and better utilization of firing fuel, all of which tend toward higher recoveries and lower operating costs.

Crude oil firing has been used at some plants to supplement the waning gas supply. At other plants coal dust firing has been tried, as have various methods for treating the residues in order to effect savings. Machines for charging and discharging retorts have been experimented with at various works, in some of which they have been pronounced a success. Improved gas producers have been installed at other plants where it is claimed they have effected a material reduction in the fuel.

The Hegeler furnace is still the standard roaster where acid is manufactured from the sulphur driven off during the roasting process.

The Grasselli Chemical Company erected a Spirlet furnace last year. This a multiple-hearth, turret-furnace, in which the hearths themselves revolve. It is one of the most popular European furnaces and it is claimed that it desulphurizes the blende with a very low fuel consumption.

The American Zinc and Chemical Company have installed Hegeler roasting furnaces at Langeloth, Pennsylvania.

Several hydro-metallurgical processes have been experimented with during the past year. At Trail, British Columbia, and Bully Hill, California, the idea is to sulphatize the zinc and electrolize the solution. So far the process is regarded as promising. The Bunker Hill and Sullivan took over the Malm process and have tested it in a 50-ton plant at Kellogg, Idaho. In this process the zinc and lead in the ore are chlorinated, then dissolved, and the metallic lead precipitated by means of metallic zinc. The zinc chloride solution is then boiled down and the zinc chloride is electrolized, giving chloride and zinc, both of which are again used in the process, except the surplus zinc which is ready for the market as spelter. While the results are encouraging some details of the process remain to be worked out.

Experiments have also been made with the Reed process for making electrolytic zinc out of bag-house fume from the Mammoth Copper Company. Experimental work at Nelson, British Columbia, has been discontinued, it having been proven that an electric zinc smelting furnace as small as one ton daily capacity is not commercially possible. The satisfactory development of a large furnace was considered too doubtful to be undertaken at Nelson.

W. McA. Johnson continued his work at Hartford, Connecticut, and organized a company to build a commercial plant at Keokuk, Iowa. He planned to use a larger furnace than any so far attempted.

The main advance in preparation of ore for market has been in the rapidly extended use of flotation for such ores as are not amenable to ordinary methods of concentration and separation. We have much to learn regarding cause and effect in flotation. No two ores seem to be affected in the same way so that each is a separate problem that must be experimented with until the proper manipulation has been determined. It is to be hoped

that someone will discover the underlying principles of this useful process. A number of modifications have been evolved, each differing in detail but all operating along similar lines.

This process will doubtless be the means of making a number of ores available that have heretofore been considered unworkable and the output of such concentrates will doubtless increase for a number of years. The product is not considered by smelters as desirable as the coarser ore. The losses are greater in transportation, in roasting and in furnacing. They likewise tend to produce a more sticky residue which it is difficult to remove from the retort, and where the charge is largely made up of such fine concentrates there has been trouble with explosions within the retort, due to the gases not being able to escape freely enough from the charge. This sometimes results in badly burning the furnace attendants.

It was formerly thought that ores with a limestone gangue were not suitable for flotation but methods have been found that are well adapted to such ores, giving a highly satisfactory product as well as recovery.

Effect of European War on Spelter Production and Consumption

The United States is not only the greatest producer but the greatest consumer of zinc in the world. Germany and Belgium rank next in production; neither equals that of the United States, but combined they produce over 40% more. Both of these great smelting centres are in the territory where active fighting is going on. Some of the works have been compelled to suspend while others have very much decreased their operations. They are cut off from a considerable portion of their ore supply which came from Australia, some 500,000 tons per year. It is likewise impossible for them to export any metal. The main district lies within 100 miles of Liège.

The zinc smelters of Upper Silesia are in the extreme southeastern portion of Prussia, mostly in the Kattowitz, Beuthen and Tarnowitz districts. They are within five or ten miles of the Polish-Russian border, near the corner of Russia, Austria and Germany and are adjacent to one another.

The zinc smelters in France are close to the territory where military operations are active and are therefore likely to be interfered with. The zinc smelters of England are not likely to be affected but those of Austria-Hungary will doubtless suffer.

Other small smelters in Europe, Australia and Japan will probably not be affected.

In 1910 Germany had twenty-four zinc smelting plants in Upper Silesia and ten in the Rhine district of Westphalia. Great Britain had seventeen and Belgium thirteen plants while France had only five.

In 1913 Germany and Belgium produced 480,816 metric tons. They consumed only 308,400 tons, leaving a surplus of production over consumption of 172,416 metric tons. The same year France, Spain, Great Britain, Russia, Austria-Hungary, Italy and several other small producing countries produced 174,486 tons. The consumption in these countries during the same period was 387,200 tons. The consumption was therefore 212,714 tons greater than the production. During this period Holland and the United States produced 344,606 metric tons and consumed 317,300 tons, leaving a surplus of 27,306 tons.

From the above it will appear that the second group of countries depended upon Germany, Belgium and Holland to supply their deficiencies. From this they are now cut off and for several months have been purchasing heavily in this country, with the result that they have absorbed the 60,000 to 70,000 tons of surplus metal largely in smelters' hands August 1st, 1914. The markets are now bare with little metal for spot delivery. The market is thoroughly disorganized and prices within a month have advanced $2\frac{1}{2}$ cents per lb., metal today being quoted at $8.50 St. Louis.

The greatest demand from abroad is for high grade and special metals for the manufacture of brass for ammunition but a large tonnage of Prime Western spelter has likewise been exported.

The galvanizing of iron sheets is dominated by Great Britain which country controls 87% of the total European export trade. Great Britain imported, in 1913, 150,000 tons of spelter used chiefly in her enormous output of galvanized iron. It would seem as though the United States would be called upon

to supply the foreign shortage of approximately 200,000 tons of spelter per year so long as the war lasts.

High markets will doubtless stimulate production in the countries not affected by the war. On the other hand, the high prices that are sure to prevail will to some extent curtail consumption, substitutes, especially for galvanizing, being used.

The galvanizing industry in this country will also be affected, where two-thirds of the spelter produced is used for this purpose, while about 25% is used in the brass industry and the remaining 8% for other purposes.

It would therefore seem as though we must expect an abnormally high spelter market while the war lasts. There will doubtless be fluctuations and it cannot now be foreseen what the ultimate effect will be.

The opening of the Panama Canal and the establishment of transportation lines from this country to South America, Australia and the Orient, combined with the present crisis, should make the introduction of American spelter into those countries much more rapid than would have been possible under normal conditions. After the war is ended the Continental smelters will begin to operate their plants and compete strongly for the trade they formerly held, but conditions are likely to be very different from those which existed prior to the war. Labour and fuel will probably both be higher. Some of the less favourably located plants may not resume, so that the European smelters are not likely to enjoy the advantageous position they occupied as regards this country. It should therefore mean a widened market and permanently increased demand for American spelter, as doubtless some of the trade secured under present conditions will become permanent.

Already trial shipments of Australian concentrates are being sent to this country and if the experiments prove satisfactory and our smelters succeed in obtaining good recoveries, at least a portion of the 500,000 tons per year formerly going to Germany and Belgium will find its way to this side unless ocean freights become prohibitive. This will result in a wider margin between the price of ore and the price of spelter than has existed for some years past.

THE POTENTIALITIES OF ZINC RESOURCES IN CANADA

The principal deposits of zinc ore in Canada are in the Kootenay district of British Columbia. The ore is a sulphide, zinc blende, and is associated with other minerals. In some few instances the zinc exists in sufficient quantity to class the deposit as a zinc deposit, although in most instances the silver-lead ore is of primary importance, the zinc being a by-product. This is especially true of the Slocan.

It is possible to occasionally hand-sort out some high grade blende, but this is rare. The Lucky Jim mine was a notable instance but even in that mine the tonnage of such ore was limited.

With a better understanding and wider application of flotation, certain ores that have in the past been neglected will become available.

The possibilities of establishing zinc smelters in Canada have been very exhaustively treated in the "Report of the Commission Appointed to Investigate the Zinc Resources of British Columbia, and the Conditions Affecting Their Exploitation" published in 1906. There has not been sufficient change in conditions, or advancement in the art, to make the proposition look more attractive than when that report was written.

While zinc smelting is apparently a simple metallurgical process, practically it is one of the most difficult. It requires skilled labour, which comes only with long experience. The individual equation of the furnace man is a more important factor than in any other metallurgical operation. Suitable and cheap fuel are essential under present methods; likewise suitable refractory clay for making retorts, as well as a market with favourable freight rates.

The consumption of zinc in Canada is comparatively small. About half of it is in the form of spelter, the remainder in sheets and rolled plate. Foreign markets must therefore be sought if any tonnage is produced; but on account of the relatively high tenor of the zinc concentates that can now be produced, together with the lower freight rate on ore than upon metal, it is more advantageous to export the ore than the metal.

Under the present favourable tariff in the United States that country would be the logical market for zinc concentrates produced in Canada. The unfavourable effect of any increase in the tariff would be minimized if the American smelters are able to retain any considerable portion of the foreign trade which they are building up under present conditions, as the ore could be brought in in bond, smelted, and the spelter exported without being obliged to pay the duty.

When electric smelting of zinc ores in the United States has advanced sufficiently to become a commercial process it may be advisable to consider smelting in Canada if it is possible to obtain cheap electric power. This would be especially true of complex, fine-grained ores containing precious metal values.

In my opinion it would not be wise to consider erecting a zinc smelter in Canada at present. It would require time to establish a successful business, which, in the meanwhile, would be accompanied by financial losses and disappointments.

In this paper I have confined myself to the production of spelter. There are, however, other products of zinc which have assumed no mean proportions in this country.

Zinc Dust

This is a fine powder formed in the condensers of the spelter furnaces when the zinc vapors are cooled too rapidly, or diluted by too large a volume of gases. Ordinarily this amounts to from 3 to 10% of the reduced zinc. The powder is metallic zinc containing about 10% oxide of zinc, which probably surrounds the metallic particles as a coating, or film. Merchantable zinc dust must not contain over 10% of oxide of zinc and must pass through a 100-mesh sieve.

In 1903 G. G. Converse and A. B. de Saulles of South Bethlehem, Pennsylvania, patented a furnace for making zinc dust, or blue powder, as it is more commonly called. This consisted of a muffle distilling furnace connected to a collecting and expansive chamber immediately adjacent to the muffles and

capable of being so regulated as to precipitate the metallic vapours therein in their substantial entirety as *zinc dust.*

Under the name of "indigo auxiliary" a paste of zinc dust is used to discharge locally the colour in dyeing cotton goods, giving a white pattern on a background of dye. Zinc dust is used in fireworks and in making hydrogen for balloons. It is the basis of various methods of dry galvanizing.

One of the most important uses of zinc dust is as a substitute for zinc shavings in zinc precipitation in the cyanide treatment of gold and silver ores. It is quicker, more effective and more convenient than zinc shavings and is usually about one-third cheaper. Heretofore, in American zinc smelting practice, the zinc dust has been charged back into the retort, as under the tariff it could not compete with the imported article. It was therefore more profitable to turn it into spelter. Several plants are at present producing zinc dust on account of restricted shipments from abroad.

ZINC PIGMENTS

There are four white pigments having zinc as a base, either in whole or in part. Zinc oxide, leaded-zinc oxide, zinc-lead oxide and lithopone. Of these, zinc oxide is the most important. In Europe the greater part of the zinc oxide is made from metallic zinc. In the United States it is made almost entirely from ores. In 1913 this country exported 14,467 short tons. The production of zinc-lead and leaded-zinc oxide is very small as compared with the straight oxide of zinc. The zinc contents of the oxide of zinc made from ore in the United States in 1913 amounted to 67,700 short tons.

Lithopone is an intimate mixture of zinc sulphide and barium sulphate, obtained by mixing solutions of zinc sulphate and barium sulphide, which precipitates all of the contained salts. The production of lithopone in the United States in 1913 was 29,685 short tons. In addition to this there were imported 2533 tons. In 1902 the production was 2250 short tons and the imports, 750 tons. During the twelve years there was an increase in the production of 1219% and 238% in importations.

Discussion

Mr. J. A. Van Mater.—As some may not be familiar with the metallurgy of zinc I may say that while the operation is usually referred to as zinc smelting this is a misnomer, as it is not a smelting operation but rather one of distillation and in order to liberate the zinc content of the ore it is necessary to have carbon in intimate contact with the particles of ore in order to reduce the zinc. This was accidentally discovered by Dony in his early experiments to which reference is made in my paper. The ore is first ground to between one and two millimetre size, mixed with suitable coal, low in volatile matter and charged into fireclay retorts. These retorts are so arranged in a furnace that the heat is externally applied, the fuel being either coal burned upon a grate at the bottom of the furnace or else fired by natural, or producer, gas, depending upon the location of the works. Clay condensers are luted into the open end of the fireclay retorts which project through the front of the furnace. The function of these is to collect the metallic vapours expelled from the ore. The accumulated metal is usually drawn from the condensers three times during the twenty-four hours. In the morning the retorts are charged with ore and coal and the furnace gradually heated up, the highest temperature usually being applied near the end of the period. Early the next morning after the zinc has been expelled, the residues are withdrawn and the retorts cleaned out when the furnace is ready for recharging. Thus each charge is worked off in a twenty-four hour period.

Mr. S. S. Fowler.—Concerning Mr. Van Mater's allusion to the possibilities of zinc-smelting by standard methods in British Columbia or the adjacent Alberta country, I regret to say that there has been no material improvement in conditions since the time of the "Zinc Commission report" of 1908. In that year, reduction works were established by French capital at Frank, Alberta, for the treatment, essentially of British Columbia zinc ores. The works were put in operation about the middle of the year; but only 60 or 70 tons of spelter was made before it became evident that the cost of operation was quite too high to permit a reasonable, if any, profit either then or in the near future. The plant was consequently closed and subse-

quently dismantled and sold for other purposes. So far as zinc-mining in British Columbia is concerned, operations for the year 1914 resulted in a slight increase of production as compared with 1913, and was larger than for several preceding years, the metal content of shipments apparently being a little more than 3,500 tons. Most of the shipments were in the form of concentrate as a by-product of the dressing of silver-lead ores, and a considerable part of the value occurs in the silver content. This last metal, in fact, has made shipments even as far as Oklahoma smelteries, not only possible but worth while, provided that the material be regarded as a by-product. It is doubtful, however, if with normal prices for zinc and without the value of the silver, zinc-ore mining and dressing *per se* can be carried on at a considerable profit at present.

MR. F. A. JORDAN.—Mr. Van Mater has suggested that because of labour conditions obtaining in Europe since the outbreak of war, the price of spelter has been maintained at a high level. Now zinc is not one of the absolutely necessary metals and it is a known fact that directly its price is raised beyond a certain figure some other metal is employed in substitution. This was the case in 1905, when the price of zinc was abnormally high.

From a geological standpoint, at any rate, it is interesting to note that some veins, the ore of which carries a very considerable quantity of zinc, have been found to occur north of Sudbury. They are in the iron formation proper, and parallel the strike of the deposits. In some cases the widths of these veins are as great as fourteen feet, but they will average, possibly, six feet.

MR. ARTHUR A. COLE.—May I ask Mr. Van Mater for some information on the subject of zinc dust production? Heretofore the supply of zinc dust for the Porcupine mills has been obtained from Belgium. That source has, of course, been cut off. The supply is now derived from the United States, and the price has been increased from seven to about twelve cents, while the percentage of metallic zinc contained in the material so supplied is decidedly lower than in the Belgium or Silesian product. In fact, the Porcupine consumers consider that, having regard to

this inferiority in quality, the price now being paid for zinc dust imported from the United States is equivalent to double that paid prior to the war.

Mr. Van Mater.—Zinc dust in the United States is produced as a by-product from the manufacture of spelter. It is collected in prolongs attached to the outer end of the condensers in which the metal collects. If the metallic vapours become too much oxidized the zinc dust is of correspondingly lower metal content. Ordinarily it is more profitable to charge the zinc dust back into the spelter furnace than to dispose of it as dust. Therefore the manufacturers of spelter have heretofore not taken the necessary precautions to prevent undue oxidation of the metallic vapours. I know of only one concern making zinc dust as a primary product. An especially designed furnace is used for this purpose. The product is very high in metallic zinc and very pure but as it is low in lead those who use it in connection with the cyanide process say that it would be better for their purpose if it contained more lead, as a small per cent of lead makes it more effective. A higher lead content can be obtained by using an ore containing more lead. There is absolutely no spelter to be had for prompt delivery, except in small quantities, even thougg the price is exceedingly high. Therefore, unless the manufacturers can get considerably more for the zinc dust than for the same metallic content as spelter they will not take the trouble to prepare the dust. Every zinc smelter is endeavouring to turn out the highest possible tonnage of spelter while present prices hold. This will tend to gradually supply the demand and when this point is finally reached we may expect a radical drop in price. A very small surplus tonnage is likely to drop the bottom out of the market. Of all metals spelter is the most sensitive to the law of supply and demand. Upon the termination of the war it is likely that the demand for zinc dust will again be supplied largely from abroad.

DEVELOPMENT OF DREDGING IN THE YUKON TERRITORY

By O. B. Perry, New York

Annual Meeting, Toronto, 1915

The first dredge in the Yukon was a small, $3\frac{1}{4}$ ft. bucket, New Zealand type dredge, built on the Lewes river in 1899. In 1901 it was moved and rebuilt on No. 42 below on Bonanza, and, in 1903, moved to the Discovery Group. It was this dredge, which was practically obsolete by 1905, and entirely unsuited for the work, that proved the practicability of dredging on the Klondike creeks.

Yukon Break-up, May 7th, 1907.

The next dredge to be built was the Canadian Number One, a 7-foot dredge which began operation in August, 1905, in the valley of the Klondike River, opposite the mouth of Bear Creek. These two dredges may be said to have shown the way to the large development which followed.[1]

[1]The operations of these dredges were described by Mr. T. A. Rickard, *Mining & Scientific Press*, August 12th, 1908.

The first three dredges of the Yukon Gold Company were built on Bonanza in 1906 and began operation early in 1907. These were followed in 1907 and 1908 by four 7-foot bucket dredges, two located on Bonanza and two on Hunker Creek. In 1910 the Canadian Klondyke Company built the first of its 16 ft. dredges, which began operation in November of that year. In 1911 the Yukon Gold Company built two all steel dredges, each of the 7 ft. size. The Canadian Klondyke Company built

The Power Plant Flume.

in 1913 Numbers 3 and 4, both 16 ft. machines. This comprises the dredging equipment now in use in the Yukon in the Klondike field. In 1907 two Risdon dredges were built on Walker's Fork, one of which was later moved to Miller Creek, where it is now operating, and a dredge, which has since been abandoned, was installed on the Bonanza Basin.

In order to operate successfully, and within the limit of expense, electric power was a necessary adjunct to the dredging. In 1906, the Yukon Gold Company installed a 2,000 K. W.

hydro-electric plant, located at the junction of the Twelve Mile and Little Twelve Mile Rivers, 32 miles from Dawson. The Granville Power Company followed four years later with a plant, located on the North fork of the Klondike River. The Canadian Klondyke Company takes its power from this plant.

In terms of investment, the Yukon Gold Company's first seven dredges cost $1,000,000. The power plant cost $550,000, and the two all steel dredges $500,000 more; making a total in-

Yukon Con. Gold Fields Co.'s Dredge 90, on Bonanza Creek, August, 1907.

vestment in dredging equipment alone of over $2,000,000. The three large dredges of the Canadian Klondyke Company are estimated to have cost $475,000 each. The power plant of the Granville Power Company cost over $1,000,000. These figures do not take into account investment in shops, buildings, and accessory plant.

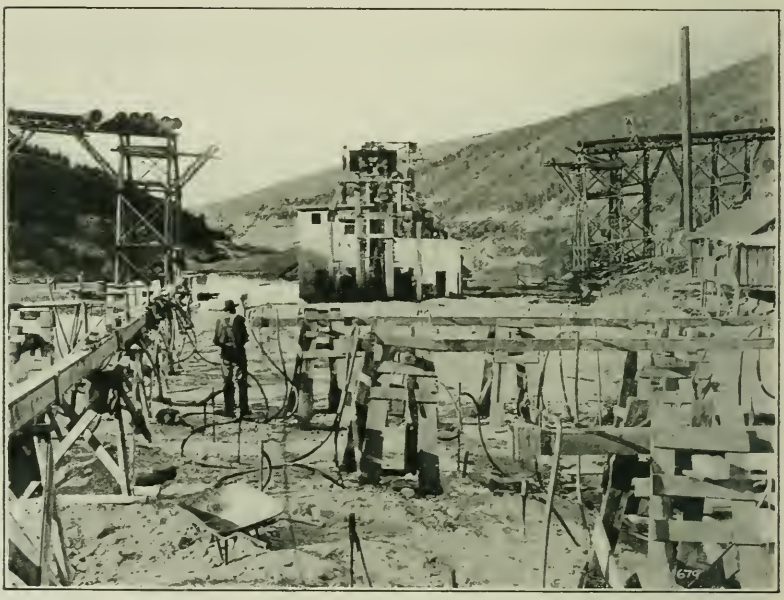
Dredging in the Yukon is now being conducted on two classes of deposits, river valley, and creek beds. The deposit

in the valley of the Klondike River is, where thawed, well adapted for dredging. The material is free wash gravel rounded and clean, averaging from 30 to 40 feet in depth. The bed rock is sufficiently decomposed to be dug with ease. The top burden, in the thawed portion of the deposit, has been largely removed by stream action and presents little or no difficulty to dredging. The creek deposits present altogether different characteristics. The deposits are shallow, for the most part, ranging about 18

Thawing Plant.—The illustration shows the points partly driven.

to 20 feet to bed rock.[1] The gravel is angular and contains very little sand. The gold values are confined almost entirely to bed rock. The creek gravels are overlaid by a deposit known locally as 'muck,' which is made up of organic matter, fine sand, and water; almost invariably frozen, and, where undisturbed, covered with a blanket of moss. The depth of the covering of muck is from 4 to 8 feet, with an average of about 6 feet. In the dredging operation on the creeks it is necessary to take up from 4 to 6 feet of the bed rock itself, which is a sericite schist and usually massive and blocky in character.

In order to make a success of a dredging enterprise so far from the source of supply, the equipment must be complete and of the best possible kind. A single breakdown, or loss of a part which could not be replaced might mean the loss of a season's work, so that the supply of extra parts and the shop equipment for repairs is much greater than ordinarily required. Generally, speaking, the dredging equipment does not differ greatly from the equipment in use in California. The machines follow the



Thawing Plant for Dredge No. 5.—Bonanza Pipe-line Trestle in the background.

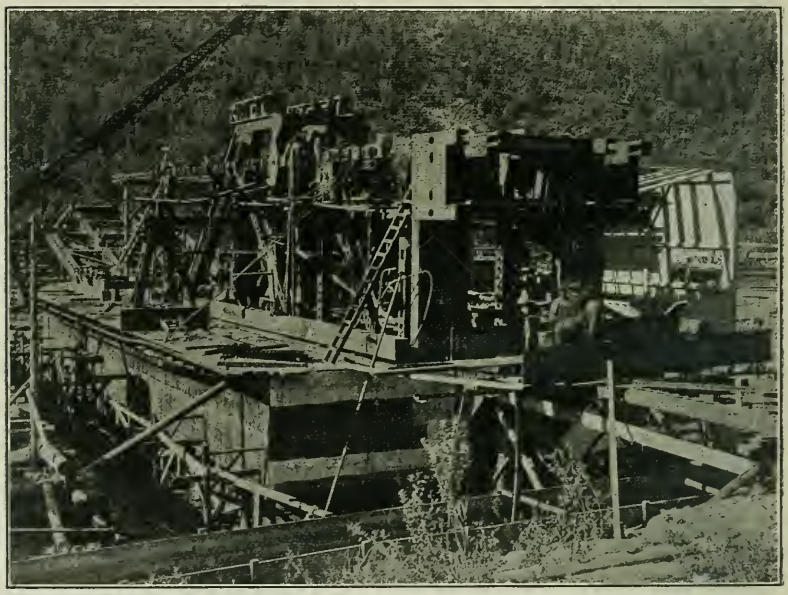
lines of the dredges developed in California, but, in the building were increased in strength of parts over anything of the size previously built. The service is extremely severe and the season short, so that all digging and working parts subjected to strain are reinforced and made heavier than is usual in dredging practice. The very best of material is used; all important shafting is nickel steel, the buckets are of chrome nickel steel, or of manganese steel. The pins are of high carbon steel, nickel steel, or chrome nickel. The tendency has been to continually increase

the strength, and, therefore, the weight of the principal parts. The last two dredges built by the Yukon Gold Company of all steel construction are 20% heavier than anything of the size in use at the present time.

Steel hull and superstructure of Dredge No. 8, in builder's yard before being "knocked down" for shipment.

Operations in the Yukon district are subject to difficulties which are not met elsewhere, except in Alaska. These may be described under two heads:

(1) *The Climate.*—The extreme cold of the northern winter limits the operation to an open season of about six months. Dredging cannot be begun until there is sufficient thaw to produce water for floatation and to clear the ponds of the winter ice, which forms to a depth of from 5 to 7 feet. Both the start and finish of the seasonal operation is hindered by cold. All moving parts, such as the tailings conveyor must be housed in and warmed by steam coils. Also, steam is used freely to keep the ladder and



Stern view of steel dredge in course of construction.

bucket line clear and to keep the gold tables in good condition. The dredging operation is terminated when the creeks freeze to such an extent that water for floatation is no longer available.

(2) *Character of Material.*—The creek deposits present many obstacles to dredging. In the first place, the deposits are shallow, which causes many moves of the dredge, both 'stepping ahead,' as the operation is called, and side moves across the pond. The shallow ground also increases the labour for outside work, such as setting 'dead men,' removing stumps, etc. The

muck itself, if thawed, is not difficult to handle, particularly if sandy in character, but it often carries, buried in it, stumps, roots, and other material which tend to choke the feed hoppers and cause delays. Digging the bedrock in order to recover the values is the most difficult part of the process. It is slow, hard work that entails heavy wear and tear on the entire machine and causes continual replacement of all digging parts. The greatest obstacle to dredging is the frozen condition of a large percentage of the creek gravels, which condition must be removed before dredging operations can be successfully conducted.

The short season during which it is possible to operate has led to many departures from usual methods. The determined effort is to get all that is possible out of the dredges in the short time allowed, and the result is a remarkably high efficiency for the time the machines are operated. For example, last season the dredges of the Yukon Gold Company averaged 87.2 per cent of the possible running time, as compared to a fair California average of 78 per cent. Each season has shown a gain in yardage per dredge hour over the season previous, until the present rate is higher in most cases than that of dredges of the same size in California where conditions are much more favourable.

The California practice is to run the dredges continuously until they get to the point of requiring a complete overhauling when they are shut down for ten days or two weeks and the necessary repairs made. In the northern practice this is changed. All the heavy repair work is done in the closed season. In the Fall, when work is suspended, the dredges are stripped of their bucket lines and conveyors. The buckets, screens, pumps, and all parts subjected to heavy wear, are taken to the shops and thoroughly overhauled. The dredges are put in a state of as nearly perfect repair as possible before the season opens and then operated with a view to keeping them going throughout the season without any shutdown.

During the operating season all repairing upon the dredge is minimized as much as possible. Instead of replacing worn bucket lips on the dredge a few at a time, as is often done, the entire bucket line is taken off and replaced by another line which has been re-lipped and entirely overhauled. This operation re-

quires only a few hours. The worn buckets are then hauled to the shop, repaired and kept on hand for the next replacement. If the distance to the shop is too great, the buckets are repaired on the ground by a shop crew. Portable compressors and special tools are provided for this work. The same method is used with lower tumblers and other parts exposed to heavy wear. When the lower tumbler plates become worn, instead of replacing the plates, the tumbler itself is removed and a newly repaired tum-

No. 9 Steel Dredge in course of construction.

bler substituted. The old tumbler is then sent to the shop or repaired and re-plated on the ground, as the case may be. Each Fall all belting is taken off and sent to warm storage, or replaced if worn.

Many detail changes have been made in the dredge machinery. The buckets have been re-designed several times since operations began. Some parts have been found to be inadequate for the heavy work. The digging ladders have all been reinforced and heavily shod on the bottom; screens re-

built; pump capacity increased; sluices and gold saving apparatus changed and improved.

The closest possible supervision is kept over the running time and performance of the dredges. Each dredge is connected by telephone with the dredge office and reports its running time and stoppages daily. The superintendence is so arranged that the Superintendent or his assistant visits each dredge at least once per day, and is on hand for any extra heavy repair jobs. In case of a shutdown for repairs, if only for a few hours, the crews are doubled up and shop men sent out for any special work. Electrical repairmen are on hand day and night for any failure or trouble in the electrical apparatus.

As a further stimulus to increasing the dredging rate, a bonus system was tried during the season of 1912, but abandoned. The conditions varied so widely from one dredge to another, and the dredges being of a different age and in different condition made it difficult to arrive at a satisfactory standard of measurement. The bonus did not act as a stimulus, but rather aroused antagonism. The operators are all picked men, who get higher pay than they receive on other dredge operations, and a trip north, which they like. No further incentive seems necessary for the men to give the best that is in them.

As illustrating the improvement in dredging in the Klondike since its inception, dredge No. 5 of the Yukon Gold Company dredged last season through the tailings of the old Lewes River dredge and make a good profit. This dredge took up 2 to 4 feet of bed rock which the old dredge was unable to handle and recovered good values in ground that had been worked by hand by open cut methods before being dredged by the small dredge.

While the short season and severe service has led to numerous improvements in detail and to increases in strength and size of the dredges and continues to offer problems which the dredge operator must meet, the greatest development in the Yukon as compared to the ordinary dredging operation has come in the handling of frozen ground. The heavy bedrock encountered on the creeks and the necessity of digging it to a considerable depth to recover the values was in itself a serious matter. No experienced dredge operator, in the beginning, would have

thought that any dredge could handle the bedrock which the machines are digging today with comparative ease. When the business was further complicated by having large areas solidly frozen, the problem of dredging such areas at a profit became one of extreme difficulty. The results of attempts to dredge partly thawed areas, and frozen bedrock, were enough to show that the frozen condition must be completely changed before dredging could be successful. Necessity compelled the adoption of the present method of thawing, which is the development, on a wholesale plan, of the method which was in use in the drift

Dredge No. 1 and Thawing Plant.

mines in the Klondike before dredges were thought of. Preparing the ground for dredging by thawing with steam was an entirely new departure in dredge mining, and the success of the method has made millions of yards of gravel available for dredging which could not otherwise be worked at a profit.

The steam point as originally used was a short piece of 3 8″ pipe drawn down at one end to a point with an orifice left for the steam to escape; the other, or head end, had a nipple welded to it for steam hose connection. The point, with steam turned on, was gradually driven into the face of the drift until driven home, then allowed to steam until the bed rock and gravel was thawed to the height of the drift. From two to six points were used in each breast, spaced from a foot to two feet apart. This simple

operation contained the essential features of the method whereby over three million yards of gravel are now being thawed in a season.

In the thawing operation for dredging the points are driven from the surface down through muck and gravel and into the bedrock from four to six feet. The points were gradually lengthened out in practice to, first 24 feet, then 30, and last season were being successfully driven 40 feet to bedrock. The points are made up of triple extra strong hydraulic pipe with drop forged head and tool steel tip. The small boilers first used have been replaced by larger sizes until the boiler equipment now comprises eight plants with a total boiler horsepower of over 2,000. The 150 horsepower locomotive type has been adopted as the unit, and the plants usually consist of a pair, or, sometimes, three boilers of about 150 boiler horsepower each.

The preliminary work for each season takes the following form: First, the dredgable area ahead of each boat is barred for frost. From the bar hole results the naturally thawed and frozen areas are blocked out. The bar tests are checked by drilling where thought necessary. Second, surveyors pick up the thawed areas which are mapped and planimeter measurements made of the frozen areas. Third, the estimated course of the boat is laid out for the year and its probable position calculated for each month of the season. Fourth, the total square yards of frozen ground which the dredge will dig in a season is measured from the map on which the season's work is laid out and the necessary wood for thawing is determined and its location decided upon.

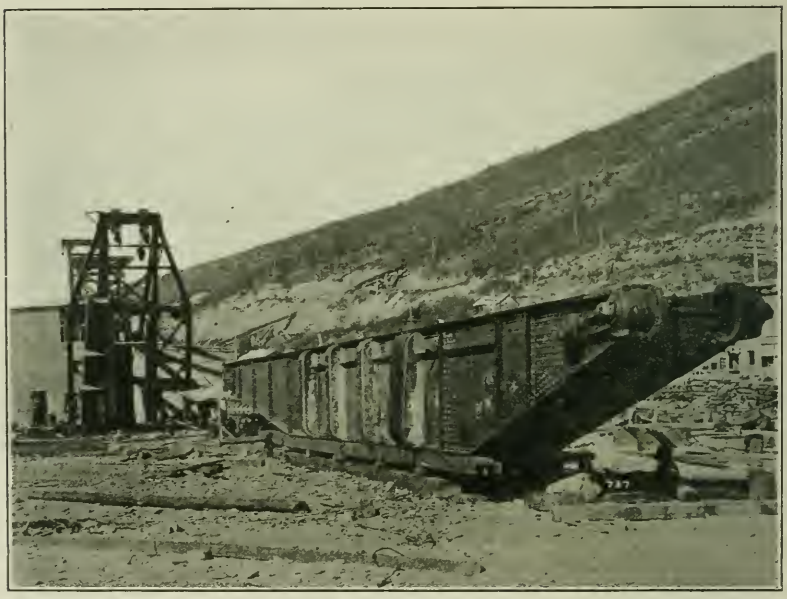
The 'sweating' of the naturally thawed ground must also be taken into account. The duty per cord, both in thawing and in sweating, to dispose of the surface frost, has been determined by experience. Knowing the square yards of frozen ground to be thawed and the area of naturally thawed ground to be 'sweated' and the duty per cord of fuel, the total fuel requirement for the season is a simple computation.

The thawing operation has been described elsewhere,[1] so

[1]T. A. Rickard, *Mining & Scientific* Press, Sept. 12, 1908.

that I need only give the sequence of events in a normal operation:

1. Steam line boxes are strung out by horses if they are available, otherwise labour is used.
2. Gooseneck boxes and headers coupled on.
3. Bar holes for starters put down.
4. Starters used to thaw holes about 8′ deep.
5. Starters pulled and long points placed in starter holes.
6. Points driven down through gravel and into bed rock.



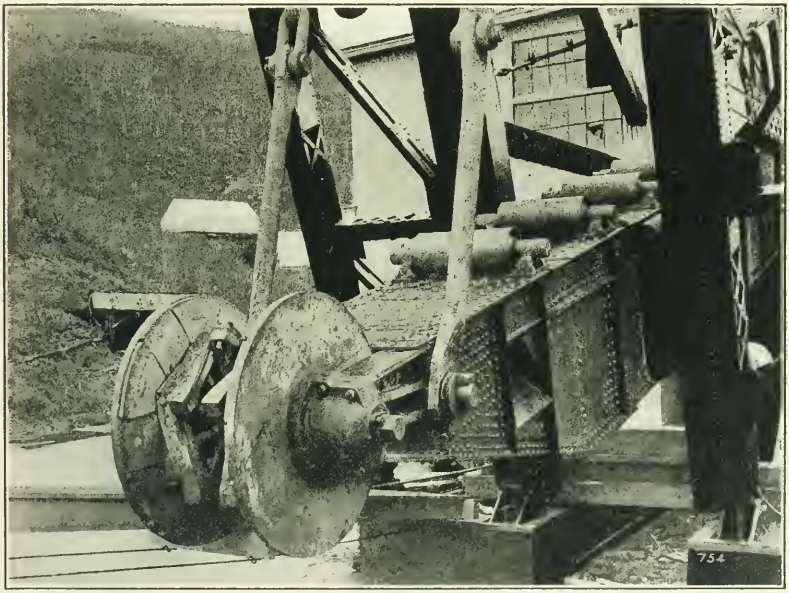
Digging Ladder.—Dredge No. 8.

7. Points allowed to steam in bed rock from 12 to 48 hours, depending on the formation.
8. Points pulled, cleaned and straightened.
9. Header moved and starters put down and cycle begins again.

The improvements that have been made in steam thawing since it was adopted for large scale dredging operations are too

many to attempt to detail in a short paper. They have taken the following general lines: First, reduction in fuel. Second, increase in efficiency of hose and points. Third, reduction in cost of driving.

The fuel has been reduced by improving the quality, reducing the radiating losses, purifying the feed water, and improving the combustion. Experiments were made with both coal and fuel oil as a substitute for wood. The points have been



Digging Ladder and Lower Tumbler.—Dredge No. 9.

greatly improved in material and manufacture and the hose has been improved in quality and its life increased. Metal hose has been used for this work with success. The cost of driving has been reduced by improvement in the method of driving, correct spacing of points, and correct pressure and amount of steam.

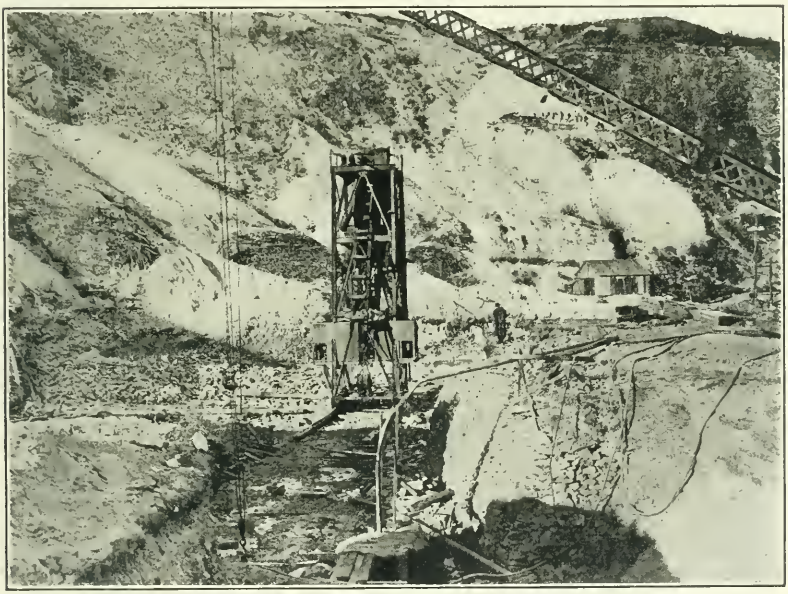
The problem of thawing has been under investigation by engineers of the Yukon Gold Company ever since the beginning of operations. In 1912 an 'efficiency' investigation was made of the operation, without, however, leading to a change in

method or producing anything new or useful. One of the engineers engaged on this work has, I believe, stated that he showed the Yukon Gold Company how to save half its thawing cost per annum, by cutting down the piping, labour and fuel consumption. Since his proposed method contemplated cutting in half the number of points driven and the amount of ground thawed, and the Company was not prepared to shut down half its dredges, his suggestions were not adopted.

Elevator Pit.—The illustration shows the blocky bedrock.

The question most often asked in connection with dredging in the north is: What does it cost to thaw and dredge frozen ground? This is a difficult question to answer on account of the wide variation in conditions. All of the creek deposits contain some areas of naturally thawed ground. The percentage of frozen and thawed areas varies on the same stream, and still wider variations occur from one stream to another. This variation reflects itself in the cost per cubic yard, the cost going up or down as the percentage of frozen ground increases or diminishes.

Another variable factor is the cost of thawing or preparing the frozen ground for dredging. Thawing on Hunker Creek, for example, is a much more difficult and expensive operation than thawing on Bonanza. In 1911, the thawing for dredge No. 4 on Hunker Creek cost $2.18 per square yard as against $1.085 ahead of No. 5 on Bonanza. The depth being practically the same, the cost of thawing per cubic yard was twice as great in the one case as in the other. Purely local conditions govern



Moving Elevator No. 3.

this cost, such as depth of ground, character of material, ease or difficulty in driving, time of steaming, cost of fuel, etc. Since the values are all, practically speaking, confined to the bedrock, the cost per square yard of bedrock thawed becomes the controlling factor in determining what ground can be dredged at a profit and what must be left.

The appended tables show the thawing and dredging cost for the past six years of the Yukon Gold Company's operation.

Table No. 4 shows the variation in working cost for the different dredges in one season. As the conditions change, the dredge holding the lowest cost for one season may show the highest cost for the next. When the costs for the eight dredges are totalled, and averaged, the results over several seasons are remarkably uniform.

While the table, Number 1, shows a gradual reduction in the thawing cost, it does not reflect the improvement in efficiency, which has been very marked. It is a very rare occurrence now for a dredge to strike a piece of ground that is not thoroughly thawed. The improvement in this direction is shown by the gain in duty per dredge hour, which increased from 162 cubic yards per hour in 1910 to 204 cubic yards per hour in 1914.

I have previously referred to the operation of the Canadian Klondyke Company in the Klondike River valley. This Company has built three 16 ft. dredges, the largest size in use in California. The dredges have operated into December, making a total working season of 270 days. The President and General Manager claims for the big dredges an operating cost of 6 cents per cubic yard. No detail has been published showing how this cost is computed. As previously set out, the Klondike River gravels, where thawed, present exceptionally good conditions for dredging. It is doubtful, however, if as low a cost as 6 cents per cubic yard can be reached if the same items are included as given in the cost figures appended hereto. In the 1913 operation of the Yukon Gold Company the items of gold tax and selling charges alone amounted to 1.98c, or practically 2 cents per cubic yard. Depreciation, taxes, insurance, etc., add another 2 cents in fixed charges, which would leave only 2 cents per yard for labour, power, repairs and supplies if a 6 cent cost were to be realized in this operation. However, the costs of the Canadian Klondike Company may be computed, they are undoubtedly low, and the Company has contributed greatly to the development of dredging in the Yukon.

The development of another method I have not touched upon, as it is, as yet, untried for any large scale operation; that is, the preparation of large areas of frozen gravel by the stripping method, or so-called 'natural thawing.' This method has been

fully dealt with in a recent article by E. E. McCarthy in the *Mining Magazine* of April, 1914. In this article Mr. McCarthy gives the results of the large scale experiments conducted by the Yukon Gold Company in 1906, 1907, 1908 and shows why the results were not such as to lead to the adoption of the method for its operations.

The dredging operation of the Yukon Gold Company has added over sixteen million dollars in gold to the wealth of the world; other companies have produced over four million, making a total of some twenty million for Yukon dredging in less than ten years of life. Few infant industries have done as well.

TABLE I

Thawing Cost, 1909 to 1914 Inclusive

Year	Square Yards				Thawing Cost in Dollars	Cost per Square Yard of Thawing in Cents
	Total	Frozen	Nat'ly Thawed	% Nat'ly Thawed		
1909	298,061	212,903	85,158	28.6	$275,112.14	155.71
1910	396,837	256,033	140,804	35.5	500,689.95	177.27
1911	555,393	432,693	122,700	22.1	696,034.69	161.66
1912	692,522	509,574	182,948	26.4	804,854.08	145.5
1913	651,505	445,624	205,881	31.6	685,570.23	154.6
1914	586,737	402,660	184,077	31.4	602,174.14	142.8

TABLE II

Dredging and Thawing Cost, 1909 to 1914, Inclusive

Year	Cu. Yds. Dredged	Total Cost of Dredging in Dollars	Total Cost per Cu. Yds. of Dredging in Cents	Total Cost Charged to Thawing in Dollars	Thawing Cost per Cu. Yd. in Cents	% of Thawing Cost of Total Cost per Cu. Yd.
1909	2,381,880	$ 760,742.25	31.94	$368,024.42	15.45	48.37
1910	3,249,788	1,010,304.89	31.09	459,437.07	14.14	45.57
1911	4,151,249	1,470,674.76	35.43	731,309.14	17.62	49.72
1912	5,157,280	1,580,289.82	30.64	774,949.47	15.02	49.03
1913	5,133,575	1,515,872.20	29.53	696,526.71	13.57	45.94
1914	4,800,781	1,326,080.75	27.62	584,760.23	12.18	44.09

TABLE III

THAWING COSTS, 1913

	Total Expenditures	Cost per Sq. Yd. in Cents	Per cent
Fixed Salaries	$ 1,614.00	0.4	.23
Labor	289,195.38	65.2	42.18
Fuel	295,928.03	66.7	43.17
Shop Expense	1,722.16	0.4	.25
Preliminary Expense	36,021.74	8.1	5.25
Material and Supplies	23,180.50	5.2	3.38
Stables	6,589.27	1.5	.96
Transportation	198.00	0.1	.03
Miscellaneous	380.33	0.1	.06
Depreciation	30,731.37	6.9	4.48
Power	9.45	....	.01
	$685,570.23	154.6	100.00

Square Yards Thawed	443,567
Number of Points Driven	63,004
Cost per Point Driven	$ 10.88

TABLE IV

THAWING AND DREDGING COSTS ALL DREDGES, 1913.

Dredge	Location	Per Cent of area Naturally Thawed	Total Cost per Cubic Yard in Cents	Thawing Cost per Cubic Yard Dredged in Cents
No. 1	Bonanza	22.1	31.93	12.55
No. 2	Bonanza	34.6	32.55	12.28
No. 3	Bonanza	14.0	33.79	18.36
No. 4	Hunker	7.2	32.11	17.95
No. 5	Bonanza	44.8	27.01	12.34
No. 6	Bonanza	18.6	33.36	16.38
No. 8	Upper Bonanza	33.7	27.92	12.97
No. 9	Eldorado	58.6	21.16	7.49

THE ECONOMIC POSSIBILITIES OF YUKON[1]

By D. D. Cairnes, Ottawa, Ont.

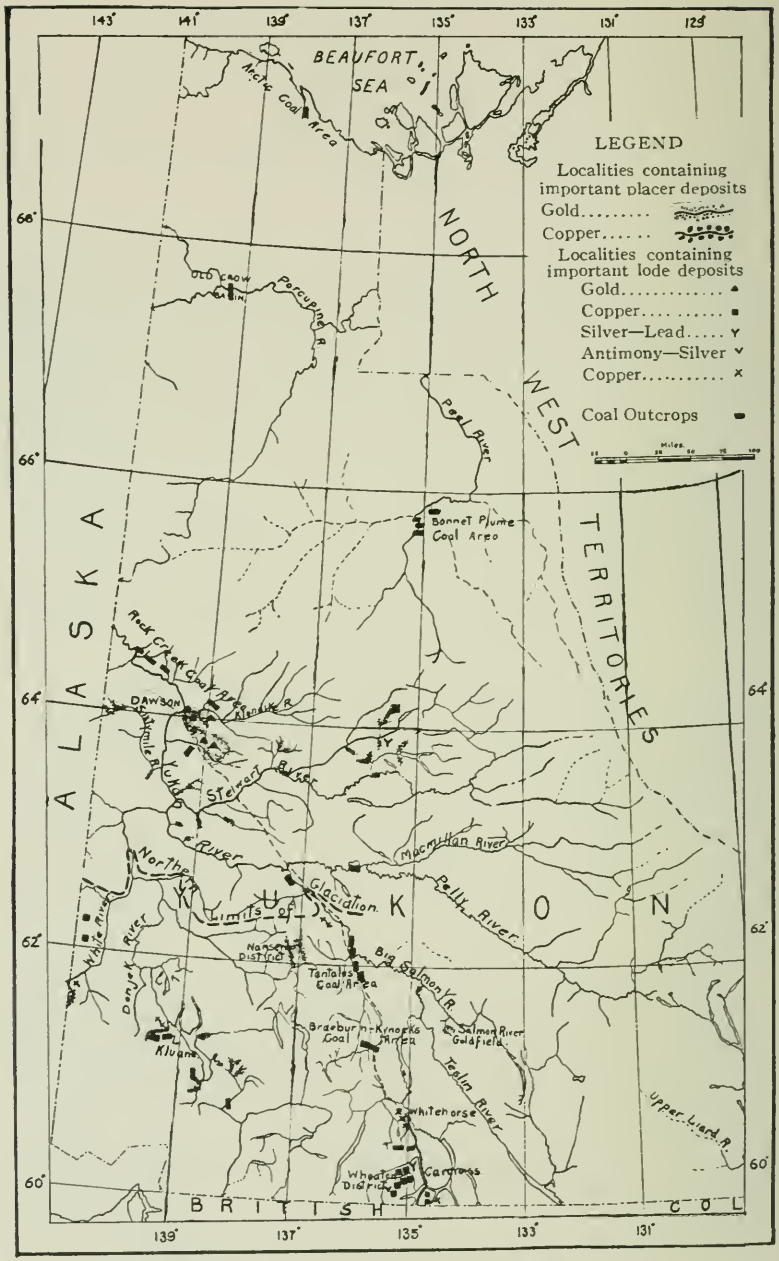
Annual Meeting, Toronto, 1915

Yukon Territory contains an area of 196,976 square miles. Thus it is considerably more extensive than the United Kingdom of Great Britain and Ireland, and is almost as large as the entire German Empire. The greater part of this great region is still practically unknown, even to the prospector, trapper or hunter: in fact almost all exploration within the territory has been restricted to areas readily accessible from the main waterways. A glance at the accompanying sketch map makes this apparent. The map shows the approximate positions of the various localities in which the more important deposits of economically valuable minerals have been found, and these are seen to be almost without exception readily accessible from the larger lakes or streams, and are in most cases in their immediate vicinity.

The writer has estimated that only about 32 per cent, or less than one-third of Yukon has been at all explored; concerning the remaining 68 per cent, or about 134,000 square miles of territory, almost nothing is known even of a general topographical or geographical nature. Also, of the 32 per cent of partly explored territory only about one half, or approximately 17 per cent of the entire Yukon, has been prospected, and of this 17 per cent, only a relatively small part has been at all closely investigated.

Up to the present, Yukon has been generally known mainly on account of its mineral resources, and particularly on account of its placer gold deposits. In addition, however, this territory has proved to be one of the most important of the fur producing sections of Canada; also agriculture is bound some day to develop into a prosperous industry. To the close of last year (1914), the placer gold production, alone, of Yukon is known to have amounted to $157,475,908 and probably considerably exceeded

[1]By permission of the Director of the Canadian Geological Survey.



Sketch Map of Yukon, showing distribution of Mineral Resources.

this amount, and all the important deposits of economic minerals, including the known valuable gold-bearing gravels, have been found within the 17 per cent of Yukon Territory that has been more or less prospected. It is thus only to be hoped that the remaining unprospected 83 per cent will prove to be relatively as valuable as the better known areas which are situated along the main waterways. This hope is strengthened when it is remembered that the discoveries so far made, dominantly occur along the main waterways, as before mentioned, and it is scarcely to be supposed that the streams of the territory in originally choosing their courses, were able to select the only areas possessing mineral wealth, and thus traverse them with a view to enabling the future prospector of to-day to make his discoveries the more readily. It must be borne in mind, however, that the prospected portion of Yukon—the 17 per cent—embraces the famous Klondike district which includes the richest gravels ever discovered in the world, and it is quite possible that no other area of similar extent may ever be found to contain gold in such phenomenal abundance. Nevertheless, the remaining portions of Yukon also contain valuable deposits of placer gold as well as other minerals, in fact, certain somewhat extensive belts are known to be quite highly mineralized. There thus seems no reason to suppose that the extensive, unexplored, interstream portions of Yukon, may not yet prove to be of great economic importance.

The principal natural resources of Yukon Territory, according to existing knowledge, include mainly mineral deposits and land suitable for agricultural and grazing purposes. In addition, the fur industry is one of considerable importance, and the forests, fish, and game are assets which must be considered. The mineral deposits are, however, of much the greatest present value; these include, chiefly, gravels containing placer gold, various types of lode deposits, and coal, as well as some native placer copper. The forest growth, though nowhere dense, is of considerable local value. The fur, fish, and game are of relatively slight importance when compared with the mineral resources, but are sufficient to largely, at least, support for many years to come, the few hundred natives in the territory, and a limited number of white men.

PLACER GOLD

The total gold production of Yukon from 1885 until the close of 1913, as shown by the statistical report of the Mines Branch of the Department of Mines, Ottawa,[1] amounts to $152,350,512, and the production for 1914 was $5,125,396, making a total of $157,475,908 to the close of 1914. This vast amount of gold has been nearly all derived from the gold-bearing gravels of the territory. A few thousand dollars has been obtained from various lode deposits, but when compared with the millions of placer gold, the lode production becomes practically negligible, particularly from a statistical point of view. Further, statistics of gold production in Yukon during the years between 1898 and 1906 are based primarily on the receipts of gold at the United States mints, and receiving offices credited to the Canadian Yukon. Thus, although a royalty was exacted on the gold output, it seems certain that considerable amounts of gold were produced which escaped royalty payment, especially during the years of high production. Since 1906, the statistics have been based on a royalty of 2½ per cent which is collected by the Department of the Interior. It would seem very probable therefore that the total gold production of Yukon has considerably exceeded the estimated $157,475,908.

Of this gold output, the Klondike district has yielded by far the greater part and Sixtymile district, Duncan Creek Mining district, or as it is more generally termed, Stewart River district, Kluane district, Salmon River goldfield, and Nansen district have contributed important amounts.

Coarse gold was discovered in Sixtymile district[2] in 1893, and from that time until the discovery of the Klondike creeks in 1896, this was the principal producing camp of Yukon Territory. It was practically abandoned in 1897 but, later, a number of miners returned and the district has each year since produced a small amount of placer gold. The total amount of gold derived from Sixtymile district is somewhat uncertain, but after visiting

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[1]Cartwright, Cosmo T., "The production of copper, gold, lead, nickel, silver, zinc, and other metals in Canada, during the calendar year 1913": Mines branch, Dept. of Mines, Can., 1914, pp. 27-29.

[2]McConnell, R. G., "Sixtymile district": Geol. Surv., Can., Sum. Report for 1901, Vol. XIV, pp. 33A-38A.

the district in 1901, Mr. McConnell writes: "The total production of the Sixtymile creeks is difficult to estimate at this late date, but was probably less than half a million dollars."[1]

The production since 1901 has been small.

Duncan Creek Mining district was examined by Mr. Joseph Keele of this Department in 1904,[2] and the principal available information of value, concerning this portion of Yukon is largely that derived from his map and report. This district has been producing placer gold since 1898, and from all available information it would appear to have yielded this precious metal to the value of between six and seven hundred thousand dollars. Mr. George P. Mackenzie, Gold Commissioner of Yukon Territory, has estimated that to the close of 1914 the district has produced about $658,000 and states that "this estimate is within $25,000 of actual production."[3]

The Salmon River goldfield has been described by Mr. McConnell[4] and was also afterwards visited by the writer.[5]

The producing creeks in this area are tributaries of the South fork of Big Salmon river, the most important of these being Livingstone creek. The total production of this area to the close of 1914, is estimated by Mr. R. C. Miller, Assistant Gold Commissioner, Whitehorse, Yukon, to have been between $750,000 and $800,000.[6]

Other authorities are of the opinion that it may have slightly exceeded this amount. In all probability the gravels of this area have yielded approximately $800,000.

Kluane district has produced a small amount of placer gold each year since 1903, but in all has yielded less than $100,000, and probably not more than $70,000. This district has also been reported upon by both Mr. McConnell[7] and the writer.[8]

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[1]Idem, p. 35A.

[2]Keele, Joseph, "The Duncan Creek Mining district": Geol. Survey Can., Ann. Report, Vol. XVI, 1904, pp. 18A-42A.

[3]Personal communication.

[4]McConnell, R. G., "Salmon River goldfield": Geol. Surv., Can., Sum. report for 1901, pp. 25A-30A.

[5]Cairnes, D. D., "Livingstone creek": Geol. Surv., Can., Sum. report for 1907, p. 14.

[6]Personal communication.

[7]McConnell, R. G., "The Kluane Mining district": Geol. Surv., Can., Sum. report for 1904, pp. 1A-18A.

[8]Cairnes, D. D., "Exploration in southwestern Yukon": Geol. Surv., Can., Sum. report for 1904; Section on Kluane district.

Plate II.—Sluicing operations on East Fork of Nansen creek, during the summer of 1914.

The production of Nansen district has probably amounted to between $5,000 and $7,000.

The total placer gold production of Yukon, excluding Klondike district, has thus probably not exceeded $2,000,000 to $2,400,000 leaving the Klondike[1] credited with between $155,000,000 and $155,500,000. Mr. McConnell spent several seasons in Klondike district[2] and his reports furnish a great part of the authentic information available concerning this portion of Yukon. He states that to the close of 1906 $119,000,000 in placer gold has been produced from the Klondike.[3]

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[1]Cairnes, D. D., "Excursions in Northern British Columbia and Yukon Territory": Guide Book No. 10, Geol. Surv., Can., 1913, pp. 95-118.

[2]McConnell, R. G., "Report on the Klondike goldfields": Geol. Surv., Can., Pt. B., Ann. Rept., Vol. XIV, 1905.

Report on the gold values in the Klondike high level gravels": Geol. Surv., Can., 1907.

[3]Idem, p. 34.

Plate III.—Placer mining on South fork of East fork of Nansen creek by underground drifting and sluicing, summer of 1914. The gravels are hoisted to surface by windlass summer and winter, and sluiced during summer months.

Between 1906 and the close of last year (1914) about $36,000,000 or slightly more than this amount was obtained from the Klondike gravels, making a total of $155,000,000 which corresponds with the writer's estimate.

The time required to deplete the Klondike gravels of their gold involves a number of questions, and is thus only very approximately known. It would seem probable, however, if the present policies and methods of the various operators be continued, that the bulk of these gravels will become exhausted as far as this can be accomplished by the most up-to-date mining methods of the present day, in 9 or 10 years: but in places where exceptionally large amounts of low-grade gravels have to be handled, as in the valley of Klondike river, possibly as much

Plate IV.—Working high level old channel gravels on Burwash creek, Klaune district, summer of 1914. Gravels cleaned off bedrock are wheeled in barrows to tipple and dumped into sluice boxes below.

as 10 years more may be required, but just how far the operators there can go with their present equipment is problematical, depending largely on the amount of thawed ground remaining unworked. It is to be expected that the mining operations and production throughout the district will gradually diminish in extent and amount as the task of exhausting the gravels of their gold content nears completion. New discoveries of more or less importance may be made, however, which will tend to somewhat lengthen the productive life of the district.

The question as to the amount of gold that will still be produced from the known gold-bearing gravels of Yukon is largely a matter of costs, and is thus also problematical. At the close of the season of 1906, Mr. McConnell estimated the future output of Klondike district, not including the Indian River

creeks to be $53,642,620, and stated that an additional production of from eight to ten million dollars might be expected from the creeks draining into Indian river, making a total from Klondike district of between sixty-one and sixty-four million dollars.[1] Since that time the district has produced approximately $36,000,000 and from all available data it would now appear that the Klondike will still produce between thirty and forty-five millions of dollars. The thirty millions checks very closely with Mr. McConnell's estimate made nine years ago. In a district as large as the Klondike, however, new methods and new developments, as well as new discoveries are always possible, and the forty-five millions is intended by the writer to cover such possibilities. These estimates as to future production have been made partly from the results of the operations and prospecting of the various mining companies, and partly from other data, all available information being employed. Since, however, the data upon which to base such an estimate is of necessity very incomplete, the estimate is consequently to be regarded as only approximate.

The future output of the other producing placer gold districts in Yukon is relatively even less definitely known than that of the Klondike, as in most localities comparatively little exploration has been conducted. The development and prospecting to date, however, would scarcely justify expectations from the explored gravels of these other now producing areas of more than from one to two millions of dollars. In case, however, certain of the old pre-Glacial channels of Kluane district and elsewhere are discovered and prove to be exploitable, the gold from this source may add greatly to this estimate.

In addition to the areas that have produced, and are still yielding placer gold, encouraging prospects are believed to have been found along the streams of a number of other localities. Among the more promising of these streams, that have so far been found, are certain of the creeks of Upper White River district,[2] as well as various tributaries of Nisling, Kluane,[3] and

[1]McConnell, R. G., "Report on the gold values in the Klondike high level gravels": Geol. Surv., Can., 1907, pp. 33, 34.

[2]Cairnes, D. D., "Upper White River district, Yukon": Geol. Surv., Can., Memoir No. 50, 1915, pp. 124-126.

[3]Cairnes, D. D., "Exploration in Southwestern Yukon": Geol. Surv., Can., Sum. report for 1914. See section on placer gold.

White rivers. Gold is also reported to have been found on Albert creek which drains into Sekulmun lake, as well as along various streams draining into Teslin lake and river, also along Big Salmon river, and on a number of the smaller tributaries of the Upper Yukon.

As concerns the future of the placer mining industry of Yukon, much is still to be expected from the as yet unprospected areas included in the vast unknown 83 per cent of the territory. Of this 134,000 square miles, certain areas or belts are especially promising from a geological point of view. In the Klondike, two main conditions appear to account for the extreme richness of the placer deposits. In the first place, the country rock or general bedrock consists dominantly of the old, probably Pre-Cambrian schists, which are much metamorphosed and highly mineralized; and secondly the district has not been glaciated, consequently whatever gold became concentrated in the stream gravels, whether old or recent, remained there until the miner arrived, and was neither swept away and scattered by glacial ice, nor became buried under boulder clay or other glacial accumulations. The old schistose rocks contain a great amount of irregularly distributed quartz, either in masses, veins, or lenses of all shapes and ranging from microscopic to several feet in thickness; and the placer gold of the Klondike originally occurred in these old schistose rocks, associated largely with the quartz. As the hills and valley walls became worn down by subaerial destructive processes, the gold originally contained in the disintegrated rock material became gradually concentrated by the streams, and was accumulated in the gravels which are being mined to-day.

In various parts of Yukon where glaciation has been effective, including Kluane and other well known districts, the valleys have been invaded by vast masses of glacial ice which has scoured the sides and floors of these depressions, and in some cases has transported and scattered the stream gravels with their gold content. Also whether or not the old gold-bearing gravels have been disturbed, the former channels with whatever of the original gravels and gold they may still have contained, became buried under vast quantities, often hundreds of feet in thickness, of boulder clay and other glacial accumulations. Thus even

where gold still remains in the various old stream channels, it is in most places difficult to find, and if discovered, may prove very expensive to mine. Thus, although the gravels of such districts may have originally been, and in places still are rich, and a certain amount of gold can even be profitably produced, still the effects of glaciation are everywhere greatly detrimental to the economic development of these areas.

On the accompanying map a line is shown marking the approximate edge of the glaciated zone. Thus, all areas outside this zone in which the older schistose rocks occur, should afford geological conditions quite similar to those existing in the Klondike, and with no more difficulties to be met in connection with the various prospecting and mining operations. There still remains to-day hundreds or even thousands of square miles of practically unprospected territory where these conditions exist. These areas are therefore considered the most promising for the occurrence of placer deposits of economic importance, and certainly warrant careful exploration.

PLACER COPPER

Native copper is found in the gravels of many of the streams distributed throughout various districts in Yukon, but Kletsan creek is the only stream along which it is known to occur in economically important amounts.[1] Kletsan creek is a small tributary of White river, which heads in Natazhat glacier in the vicinity of the 141st meridian—the Yukon-Alaska International Boundary line. The first definite information concerning this occurrence is contained in Hayes' report of a trip he made from Selkirk to Skolai pass in 1891 in company with Lieutenant Schwatka.[2]

Previous to this time the Indians had carried on quite a traffic in native copper obtained chiefly or entirely from Kletsan creek, this metal being used for arrow heads, knives, cooking utensils, and also for bullets, where lead could not be obtained.

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[1]Cairnes, D. D., "Upper White River district, Yukon": Geol. Surv., Can., Memoir 50, 1915, pp. 133-135.

[2]Hayes, C. W., "An expedition through Yukon district": Nat. Geog. Mag., Vol. IV, 1892, pp. 143-145.

In 1902, an attempt was made to test the placer copper possibilities of this locality, but on account of glacial ice and snow on the high ranges at the head of the creek, and a number of other adverse conditions, unfavourable conclusions were reached. There is undoubtedly, however, a considerable amount of native copper not only along Kletsan creek but in the gravels of a number of neighbouring streams, which may yet prove to be exploitable, when transportation facilities are improved.

LODE DEPOSITS

General Statement.—Lode deposits are widely distributed throughout the explored portions of Yukon, and embrace a considerable variety of types including gold quartz veins, gold-tellurium quartz veins, gold-silver quartz veins, antimony-silver veins, silver-lead veins, copper veins, and contact-metamorphic deposits which are mainly of importance for the copper ores they include.

The principal areas in which lode deposits of importance occur, include the following, which are mentioned in order commencing at the north and proceeding southward:—Klondike district, Stewart River district, Williams and Merritt Creeks area, Upper White River district, Kluane district, Aishihik Lake district, Whitehorse copper belt, Wheaton district, and Windy Arm district. Of these, the Klondike, Stewart River district, Aishihik Lake district, and Whitehorse copper belt, are somewhat widely separated, and are distributed throughout the Yukon plateau. Upper White River, Kluane, Wheaton, and Windy Arm districts, however, are situated along the eastern edge of the mountains of the Coastal system, and constitute portions of a well mineralized belt which appears to follow all along the inland boundary of this mountain terrane, at least throughout Northern British Columbia and Yukon. Of these numerous districts in which lode deposits are known to occur, only the Klondike, Whitehorse copper belt, and Windy Arm district have been or are producers, ore having been shipped from Whitehorse copper belt and from Windy Arm district, and a limited amount of quartz has been mined in Klondike district. As an indication of the extent of the lode mining industry in Yukon,

the number of quartz claims in good standing during the fiscal year ending March 31, 1914 aggregated 908, of which 151 were crown granted.

Klondike District.[1]—In Klondike district, quartz occurs very plentifully distributed throughout the schistose rocks which are there so extensively developed, and although the greater number of the deposits are small and non-persistent, the aggregate amount of quartz is very great. Occasional very encouraging assays are also obtained, but the average returns from all the deposits so far sampled, have been low. The quartz is practically all free-milling, and is very slightly mineralized, the only metallic constitutents apparent, being pyrite, and rare particles of magnetite, chalcopyrite, galena, and native gold.

Considerable development work has been performed in places, but only one property, the Lone Star, can in any sense be considered a producing mine. Since 1909 more or less mining and development work has been in progress on this property, and a certain amount of ore material has been treated in a small stamp mill belonging to the Lone Star Company, as a result of which, gold to the value of a few thousand dollars has been recovered. The gold obtained would not nearly pay the total cost of the mining operations and equipment to date, nevertheless it has now been demonstrated that portions of the quartz and adjoining wall rock on this property contain sufficient gold to pay for treatment. It is consequently hoped that a great amount of the similar appearing material which occurs not only on this property, but which is so extensively developed throughout the Klondike, may yet prove to be profitably exploitable. At present, however, it is uncertain whether or not any considerable tonnage of gold-bearing quartz occurs in the Klondike which can be treated at a profit under existing conditions.

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[1]Brock, R. W., "Yukon": Geol. Surv., Can., Sum. report for 1909, pp. 16-23.

Cairnes, D. D., "Quartz mining in the Klondike district": Geol. Surv., Can., Sum. report for 1911, pp. 33-40.

McLean, T. A.—"Lode mining in Yukon": Mines branch, Dept. of Mines Can., 1914, pp. 17-127.

Stewart River District.[1]—In Stewart River district, a number of promising lode deposits have been discovered on Dublin gulch, on Galena creek, and elsewhere, the more promising of which are veins of the arsenical gold-quartz, gold-silver, or silver-lead types. These veins are in places highly mineralized, and are reported to contain considerable amounts of valuable ore. Very little definite information concerning these deposits is, however, available.

Williams and Merritt Creeks.[2]—Merritt creek empties into Lewes river on its left limit, five miles below Yukon Crossing, while Williams creek joins the river one mile farther downstream. In the vicinity of these creeks, there occur a considerable number of veins of quartz impregnated with copper minerals, chiefly bornite and chalcopyrite. The veins range in thickness from a few inches or less, to five or six feet or even slightly more in thickness, and in places appear to be fairly persistent. Average samples of these veins show them to contain from less than one per cent to over four per cent copper, as well as small amounts of gold and silver, generally amounting to less than one dollar in the combined value of these two metals.

Bedrock in this locality is in most places obscured by superficial deposits, so that the discoveries that have been made are more or less accidental, and due to the overburden having been removed by some fortunate natural cause. Since so much mineralized quartz has already been found in this vicinity with so little of the rock formation exposed to view, it seems probable that were the superficial deposits removed a great amount of quartz would be revealed, and as mineralization is here so general, it is quite possible that some of the quartz found, would carry sufficient amounts of ore minerals to make it profitably exploitable.

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[1]Keele, Joseph, "The Duncan Creek Mining district": Geol. Surv., Can., Sum. report for 1904, Vol. XVI, pp. 38A-40A.

McLean, T. A., "Lode Mining in Yukon": Mines branch, Dept. of Mines, Can., 1914, pp. 127-159.

[2]Cairnes, D. D., "Williams and Merritt creeks": Geol. Surv., Can., Sum. report for 1909, pp. 57-60.

Upper White River District.[1]—Upper White River district constitutes a portion of a well mineralized general region, and possesses itself a considerable degree of mineralization. The more promising of the lode deposits that have been there discovered are gold-quartz veins, and veins and associated irregular deposits containing or consisting entirely of copper minerals mainly native copper. The individual quartz veins and masses that have been found, although in places quite large and persistent, are nevertheless only sparsely mineralized. It is quite possible, however, that deposits of similar extent and size may yet be discovered containing gold either alone or associated with other minerals, in sufficient amounts to allow of their being worked at a profit. Native copper has long been known to occur in White River basin; this metal, however, has been found in Upper White River district in economically important quantities on only one property which is known as Discovery Copper Grant. The native copper on this property is an oxidation product of chalcocite which in turn may with depth give place to chalcopyrite.

Thus the native metal cannot be expected to extend far below the surface, and it is to the primary sulphides that we must look for any extensive or persistent ore bodies. Sufficient development has not yet been performed to determine whether or not these sulphides are sufficiently concentrated to constitute economically valuable ore bodies.

Kluane District.[2]—Throughout Kluane district, lode deposits have a somewhat wide distribution, veins in which copper minerals are conspicuous, predominating. Nowhere, however, have deposits as yet been found which, under existing conditions, can be profitably exploited. It is quite possible, nevertheless, considering that the district is in general well mineralized, and has as yet been only slightly prospected, that deposits will yet be found that can be worked at a profit.

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[1]Cairnes, D. D., "Upper White River district, Yukon": Geol. Surv., Can., Memoir 50, 1915, pp. 120-126, 135-139.

[2]McConnell, R. G., "The Kluane mining district": Geol. Surv., Can., Sum. report for 1904, pp. 17A, 18A.

Cairnes, D. D., "Exploration in Southwestern Yukon": Geol. Surv., Can., Sum. report for 1914. Section on copper deposits.

Aishihik Lake District.[1]—A number of copper deposits which are generally designated as the Aishihik Lake deposits, occur in the vicinity of Giltana lake, a small body of water situated near Aishihik lake. The country rock in this vicinity is dominantly mica schist which includes in places mineralized bands, the ore minerals being chiefly magnetite and chalcopyrite. The more important of these ore bands or bodies range in thickness from 6 to 20 feet, and are exposed along the surface in places for as much as 200 feet or even greater distances. The average copper content of the deposits sampled, runs from 1.35 to 9 per cent. The deposits constitute a valuable future asset to the country, but under present conditions, could not be worked at a profit.

Whitehorse Copper Belt.[2]—Whitehorse Copper Belt is situated in the southern part of Yukon about 45 miles north of the British Columbia boundary, and extends along the western side of Lewes river for a distance of about 12 miles. Most of the important mining properties are within 4 to 7 miles of Whitehorse, the terminus of the White Pass and Yukon railway. This area has been carefully described in detail by Mr. McConnell and others so will be only briefly mentioned here.

The ore bodies are all of the contact-metamorphic type and fall into two classes:—those in which the copper minerals are associated with magnetite and hematite; and those in which the various silicates, principally garnet, augite, and tremolite are the chief gangue minerals.

The magnetite ore bodies are numerous, and occur either enclosed completely in altered limestone, along a lime-granite contact, or, in a few instances, in areas of altered granite. The largest bodies so far discovered are from 100 to 360 feet in length. These magnetite masses are sprinkled more or less plentifully throughout with grains and small masses of bornite and chalco-

[1]Cairnes, D. D., "The Giltana Lake claims": Geol. Surv., Can., Sum. report for 1908, pp. 30, 31.

[2]McConnell, R. G., "The Whitehorse Copper belt, Yukon Territory": Geol. Surv., Can., 1909.

Cairnes, D. D., "Whitehorse Copper Belt": "Excursions in Northern British Columbia and Yukon Territory": Guide Book No. 10, Geol. Surv., Can., 1913.

McLean, T. A., "Lode Mining in Yukon": Mines Branch, Dept. of Mines, Can., 1914, pp. 159-165.

Plate V.—Pueblo mine, Whitehorse Copper camp, showing surface equipment and buildings.

pyrite. The copper percentage varies greatly in different parts of each deposit, but the general average is between 3 and 4 per cent. The gold and silver are negligible in some of the ore bodies and important in others. Hematite masses are much less common than the magnetite bodies, the deposits on the Pueblo property being the only large bodies so far discovered. (Plates V, VI.) These differ from the magnetite ore bodies principally in the greater oxidation of the copper minerals. Deposits characterized by the garnet-augite-tremolite gangue are numerous wherever the lime-granite contact is exposed. They vary from low grade deposits containing only a sprinkling of copper minerals to considerable lenses of valuable ore. The valuable minerals are similar to those in the iron masses and consist mainly of bornite and chalcopyrite. On one property, bornite is absent however, and chalcopyrite is associated with mispickel. The siliceous ores contain, as a rule, a higher percentage of copper than the iron ores, those shipped up to the present time averaging probably over 8 per cent. The precious metal contents are moderate, seldom exceeding $3 per ton.

Plate VI.—Pueblo mine, Whitehorse copper belt, showing face of huge open cut, from which was obtained the ore of the first important shipments from this

Considerable development has been performed on a number of properties within this belt, but owing to the very low prices of copper during the last few years, the only property that has recently been worked or has shipped any ore is the Pueblo, which also closed at the commencement of the war. During the past three years the Altas Mining Company shipped about 90,000 tons of ore from their properties, all or nearly all of which came from the Pueblo. This is believed to have averaged 3.73%, 2.71%, and 3.80% copper from 1912, 1913 and 1914 respectively and to have contained averages for each year of from .60 to .81 ozs. in silver.

During the summer of 1907 Mr. McConnell estimated that probably half a million tons of ore was in sight in the Whitehorse copper belt as a result of the development then performed.[1] Since 1907 not over 100,000 tons of ore have been shipped out of the camp and important bodies of ore then unknown have been discovered. Thus undoubtedly when the war is over and the money stringency is less severe, and particularly if the prices of copper should rise slightly, there may be expected an important tonnage from this camp.

Wheaton District.[2]—In Wheaton district, a number of promising ore bodies occur. These include mainly gold-silver, antimony-silver, and silver-lead veins. No ore other than small test lots have been shipped from this district, but in most places very little development work has been performed. As a result of the prospecting and exploratory work to date, however, it is evident that there occurs within the district a number of ore-bodies which will some day be developed, and which could now be profitably exploited were conditions of transportation more favourable.

Windy Arm District.[3]—The area generally known as Windy Arm district extends northward from the 60th parallel, the

[1]McConnell, R. G., Op. cit., p. 3.

[2]Cairnes, D. D., "Wheaton district, Yukon Territory": Geol. Surv., Can. Memoir No. 31, 1912, pp. 85-145.

[3]McConnell, R. G., "Windy Arm district": Geol. Surv., Can., Sum. report for 1905, pp. 26-32.

Cairnes, D. D., "Report on a portion of Conrad and Whitehorse mining districts, Yukon": Geol. Survey, Can., 1908, pp. 14-18.

McLean, T. A., "Lode Mining in Yukon": Mines Branch, Dept. of Mines, Can., 1914, pp. 171-201.

British Columbia boundary, to Nares and Tagish lakes, a distance of from 10 to 12 miles, and reaches from Lake Bennett east to Windy Arm, a distance of about 10 miles. Throughout this area, a large number of quartz veins occur, which range from a few inches or less to 10 feet or even more in thickness. These occur mainly in a group of semi-basic rocks which are dominantly andesitic in character, but one vein—that on the Big Thing property—occurs in a granitic rock. These veins are composed mainly of quartz which carries quite a variety of ore minerals the chief of which is, in most places, argentiferous galena. In addition to galena, there also occur argentite, Freibergite, pyrargyrite, stephanite, tetrahedrite, native silver, native copper, lead carbonate, pyrite, arsenopyrite, chalcopyrite, zinc blende, and Jamesonite.

It is estimated that over a million dollars has been expended in this district by Col. J. H. Conrad and associates. Several long aerial tramways were constructed, a number of long crosscut tunnels were driven, a concentrating mill was erected on Windy arm, a power house was constructed on Lake Bennett, and a certain amount of actual development work was performed.

A number of promising bodies of ore occur in Windy Arm district, which will doubtless some day be exploited. In fact, except for the depressing financial circumstances due to the war, there would appear to be no reason why, even under present conditions, certain of these ore veins could not be worked, if the mining operations were conducted under skilled and careful management. There is every reason to believe that at least hundreds of thousands of tons of gold-silver ore will yet be mined in this district, and it is hoped this will be realized in the somewhat near future.

COAL[1]

The coal-bearing formations of Yukon are all of either Tertiary or Jura-Cretaceous age,—the mineral fuels in the Tertiary beds throughout the territory being lignites characterized in most places by containing considerable fossil resin or

[1]Cairnes, D. D., "The Yukon Coalfields". Trans. Can. Min. Inst., Vol. XV, 1912, pp. 364-396.

amber, while those of Jura-Cretaceous age range in character from high grade lignites to anthracites.

Tertiary coal-bearing beds do not cover very extensive areas in Yukon, but have a somewhat wide distribution, and in places, apparently, constitute remnants of once larger areas now infolded with older terranes; in most cases, however, they represent deposits laid down in separate basins of deposition. The fossil plant remains found in these beds, show that most of them at least, are of fresh water origin. These lignite-bearing Tertiary beds appear to all belong to the Kenai series[1] which is the oldest known Tertiary in Yukon and Alaska, and is generally referred to the Upper Eocene. There rocks are, in most places, but little disturbed, although locally they have suffered considerable deformation. They consist, typically of light-coloured, slightly coherent, conglomerates and sandstones and dark to light coloured, soft shales and clays. In places volcanic materials occur associated with these slightly consolidated sediments (Plate VII).

The Jura-Cretaceous sediments[2] consist mainly of conglomerates, quartzites, sandstones, graywackes, arkoses, tuffs, shales, and slates, having a wide range of colour and differing greatly in the amount of metamorphism they have suffered. In general they are considerably more indurated, and the beds have been much more disturbed than those of Tertiary age. The Jura-Cretaceous beds appear to be remnants of once more extensive areas which were originally all connected but have been reduced by erosion to their present proportion. In Southern Yukon where these beds have been most studied the uppermost member, the Tantalus conglomerate, is composed dominantly of cherty conglomerate beds which have an aggregate thickness of at least

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[1]Collier, A. J., "The coal resources of the Yukon, Alaska": Bull. U.S. Geol. Surv., No. 218, 1903, pp. 17-19.

Brooks, Alfred H., "The geography and geology of Alaska": U.S. Geol. Surv., Prof. paper No. 45, 1906, pp. 237-244.

Cairnes, D. D., "The Yukon Coalfields": Trans. Can. Min. Inst., Vol. XV, 1912, pp. 365-367.

"Exploration in Southwestern Yukon": Geol. Surv., Can., Sum. report for 1914. Section on Coal.

[2]Cairnes, D. D., "Preliminary memoir on the Lewes and Nordenskiöld Rivers coal district": Geol. Surv., Can., Memoir No. 5, 1910, pp. 30-38.

"Wheaton district, Yukon Territory": Geol. Survey, Can., Memoir No. 31, 1912, pp. 53-59.

1,000 feet. The underlying Laberge series has an average thickness of about 3,800 feet.

In the Jura-Cretaceous beds, two distinct coal horizons have been recognized. The upper horizon occurs well up in the Tantalus conglomerates, and the lower horizon is in the Laberge rocks within a zone 200 to 300 feet below the Tantalus conglomerates.

The beds found to be coal-bearing in Yukon occur in at least 19 distinct areas. In 14 of these, coal of economic importance

Plate VII.—Huge amphitheatre in Duke River coal area, Kluane district, in which at least twelve seams of coal exceeding one foot in thickness, are exposed.

has been discovered, and in the other areas, may yet be found when these have been prospected.

The following table gives the extent of these rocks:—

Extent of known Tertiary beds in Yukon	2,140	sq. miles
" Jura-Cretaceous beds in Yukon	4,110	"
Total	6,250	sq. miles
Probable extent of Tertiary beds in Yukon	4,500	sq. miles
" Jura-Cretaceous beds in Yukon	19,700	"
Total	24,200	sq. miles

At only five points in Yukon has coal actually been mined, viz.:—On Cliff creek, on Coal creek, (tributary of Yukon river), on Coal creek (tributary of Rock creek), at Five Fingers mine, and at Tantalus mine. The first three of these occur in the Rock Creek Tertiary basin, and the last two are situated within the Tantalus Jura-Cretaceous area. At two or three other points the measures have been somewhat prospected. The only two properties that have been in operation since 1908 are the Sour Dough mine on Coal creek (tributary of the Yukon) and the Tantalus mine situated on Lewes river, about midway between Whitehorse and Dawson. These properties are still operating.

The following table gives the analyses of a number of typical coals from different parts of the territory.

Locality.	Age.	Hygroscopic Water.	Volatile Combustible Matter.	Fixed Carbon.	Ash.
Cliff Creek	Tertiary	8.57	42.04	45.77	3.62
	"	10.58	40.10	46.74	2.58
Sour Dough Mine	"	17.10	34.50	38.40	10.00
	"	14.57	33.11	37.15	15.17
Coal creek (tributary of Rock creek)	"	18.31	34.96	40.88	5.85
	"	19.37	33.85	37.45	9.33
Duke River area, Kluane district	"	11.20	40.90	42.50	5.40
	"	9.80	43.90	44.70	1.60
Five Fingers mine	Jura-Cretaceous	5.95	40.46	45.16	8.43
Tantalus Butte	"	13.64	31.83	51.84	2.69
	"	12.87	31.72	49.51	5.90
Braeburn-Kynocks area	"	8.98	29.62	48.30	13.10
	"	12.02	34.28	42.56	11.14
Whitehorse area	"	2.15	6.01	69.86	21.98
	"	3.76	8.34	62.50	25.40

The following table gives the probable amount of coal in Yukon in seams 1 foot or more in thickness.

Field.	Age.	Tons, (2204 lbs.)
Whitehorse area	Jura-Cretaceous	40,000,000
Tantalus area	"	150,000,000
Braeburn-Kynocks area	"	50,000,000
Selkirk area	"	10,000,000
Pelly River areas	"	
Arctic area	"	
Rock Creek area	Tertiary	3,000,000,000
Sheep Creek area, Kluane district	"	40,000,000
Jarvis River area, Kluane district	"	
Duke River area, Kluane district	"	250,000,000
Bonnett Plume area	"	1,500,000,000
Indian River area	"	150,000,000
Old Crow basin	"	
Frances and Liard River basins	"	
Total		5,190,000,000

AGRICULTURE[1]

Grasses of various kinds grow well and even luxuriantly in certain parts of Yukon, particularly throughout the southern portion of the territory. Along many of the main lowland depressions, also, including portions of the valleys of Nordenskiöld, Hutshi, Takhini, Dezadeash, Duke, and Nisling rivers, extensive valuable meadows occurs. In fact, in most places, where the underbrush and moss have been burned off the lower hillsides, and the bottoms of the main valleys, grasses, particularly of the "redtop" varieties, spring up rapidly and thickly, growing in places to a height of over 3 feet. This rapid growth immediately after a fire is partly due to the fact that the grass roots already there remain uninjured by the fire, and furthermore, the grass seeds which may for years have accumulated over the surface, gradually work down through the moss where they are preserved, and wherever the moss is removed by fire or otherwise the grass seeds germinate at once. A luxuriant growth unobstructed by moss and underbrush thus results from both roots and seeds. These "red-top" grasses are not only excellent pasture grasses but in addition furnish a good grade of hay, and are particularly

[1]Macoun, John, "The climate and flora of the Yukon district". Geol. Surv., Can., Ann. Rept., Vol. XV, 1902-3, pp. 38A-54A.

valuable to stock wintering out, as the grass heads retain their seed which remains quite preserved and consequently highly nutritious. Thus animals can readily live on the grain heads where snow is too deep for the grass stocks and blades to be reached.

Plate VIII.—Typical stretch of meadow land near head of Nisling river.

These meadows should thus yet prove of considerable value. Nowhere perhaps were most beautiful or attractive stretches of these grass lands noted by the writer than occur along the upper portion of the south branch of Nisling river, where for several miles, a tall, fairly thick growth of grass extends everywhere over the wide valley lowlands of this stream and some of its tributaries in this neighbourhood (Plate VIII).

The following is a list of the characteristic grasses of Southern Yukon, which has been kindly prepared by Mr. J. M. Macoun of the Geological Survey, from specimens collected by the writer:—

Hierochloa alpina R. and G.

Poa crocata Rydb.

A blue grass—is one of the best pasture grasses on the prairie from Manitoba to the mountains, being there very plentiful.

Agropyrum violaceum. Hornm.

The only true "bunch" grass in the region.

Bromus Pumpellianus Scribn.
Arctogrostis latifolia Trin.
Calamagrostis purpurascens R. Br.
Calamagrostis Langsdorfii.
Calamagrostis canadensis Beauv.

The varieties of *Calamagrostis* or blue-top grasses, are the "red-top" grasses of the far northwest, and not only are they good pasture grasses, but they are the chief hay grasses of the northwest, due to their height and nutritious value.

Festuca saximontana Rybd.
Festuca altaica Trin.
Festuca rubra L.

These varieties of *Festuca* are known as the Fescue grasses, and are the most valuable pasture plants when found in sufficient quantity.

Trisetum spicatum (L) Richter

Grass suitable for horse feed is thus available in favourable localities throughout the entire year, and commencing the latter part of May or early in June becomes quite plentiful, and from then until October, pack-horses, if well cared for and not worked too hard, will in most parts of Southern Yukon, at least, subsist on what natural fodder is available. Horses also will winter out safely without artificial shelter or without being fed, if they are in good condition when winter sets in, and if they are left in suitable localities.

Throughout Southern Yukon, also, where careful gardening has been attempted, such has in most cases been attended with very gratifying results. In the vicinity of Dawson, for instance, vegetables and flowers grow luxuriously, the flower gardens being particularly beautiful and a never ceasing source of surprise to those visiting Yukon for the first time. Also from their gardens the people of Dawson are supplied with celery, rhubarb, radishes, lettuce, onions, turnips, beans ,parsnips, carrots, peas, cabbage, cauliflower, Scotch kale, and many other pot-herbs. These have all passed the experimental stage and compare very favourably with vegetables grown elsewhere, and the celery as well as perhaps other varieties, surpass in quality those from most districts farther south. Nothing is now necessary to success in growing these vegetables but care in cultivation. Potatoes quite compar-

able with those from the "outside," are also grown if care is taken in selecting the seed and if they are planted in suitable ground. At many points along Yukon and Lewes rivers, farther south, gardening is quite as successful as at Dawson, but has not been so extensively practised. Various grasses and hays have also been very successfully grown at a number of points.

During the summer of 1902 Professor John Macoun of the Geological Survey, visited Southern Yukon, and his report contains a great amount of valuable information on the climate and flora of this district.[1] He states, "There is no reason why all the oats, barley and fodder of all kinds with every vegetable required in the home should not be grown around Dawson."[2] He also adds, speaking of Southern Yukon in general: "With the facts learned last season (1902) and my former knowledge of the Peace River country, the Mackenzie River valley, and northern British Columbia, I am quite within the mark when I say that all the land having a suitable soil within this immense area will in the future produce enormous crops of all the cereals, wheat included. It is well within the memory of us all that growing wheat was for many years considered a doubtful matter at Edmonton and Little Slave Lake. These points have passed the experimental stage and now good crops of wheat are secured every year. Two factors combine to make this success. The wheat itself is gradually conforming to its environment and ripening earlier, and local frosts are becoming rarer as the land comes more under the plough. The same changes will take place farther to the north, and when wheat is grown as winter wheat and can start at once after the snow is off, it is hard to state how far this may be, at any rate as far as Dawson in latitude 64°19′ where we know there are three months without frost."[3]

The United States Department of Agriculture, also, has established experimental stations at Fairbanks and Rampart in Central Alaska where the general climatic conditions are very similar to those in portions of Southern Yukon, and at these stations it has been demonstrated beyond a doubt that farming

[1] Macoun, John, "The climate and flora of the Yukon district": Geol. Surv., Can., Ann. Rep., Vol. XV, 1902-3, pp. 38A-54A.

[2] Idem. pp. 48A, 49A.

[3] Idem, p. 53A.

can be made to pay in Alaska.[1] Referring to potatoes grown at the Fairbank station, Professor Georgeson states, "Some of the merchants who have handled the station potatoes stated that their customers would ask for them and take them in preference to potatoes from the States."[2]

Spring wheat also has been completely matured at the Fairbank station, and when spring wheat matured it follows as a matter of course that all varieties of barley and oats also matured. Alfalfa and red, white, and alsike clovers also all made satisfactory growth at Fairbanks, and seed has been matured on certain species of alfalfa, which is considered to be a very important step and of the greatest significance.[3] Among the most important works performed at these stations are the hybridizing or breeding of new varieties of grains, by crossing and selection, and the propagation and cultivation of various hay grasses, fruits and vegetables.

If similar government experimental work could be performed in Yukon it would undoubtedly be of considerable benefit to the territory, but in any case the people of Yukon may greatly profit by the work already done in Alaska, which is sufficient at least to demonstrate that eventually Southern Yukon will be of importance as an agricultural district. In certain localities where a first crop has been more or less a failure, this has unfortunately discouraged future attempts. It is now known, however, that such first crops are very liable to be unsuccessful, as the seed is in many cases unadapted to the climate, and the soil requires cultivation before really satisfactory results are attained. Thus, continued and persistant efforts are warranted, and if properly prosecuted are certain of remunerative returns. It is well known that stock raising, farming, and various agricultural pursuits are successfully conducted in Siberia and other northerly countries where the climate and general conditions are no more favourable and in some cases less so than in Southern Yukon.

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[1] See annual reports of the Alaska Agricultural Experiment stations for past few years, published by the U.S. Dept. of Agriculture, Washington, D.C.

[2] Georgeson, C. C., Ann. Rep., Alaska Agricultural Experiment Station, U.S. Dept. of Agri., Washington, D.C., 1913, p. 14.

[3] Georgeson, C. C., Ann. Rep., Alaska Agricultural Experiment stations, U.S. Dept. of Agri., Washington, D.C., 1913, pp. 17, 18.

At present, however, while there is still considerable free land open to homesteaders in the Northwest territories and in parts of British Columbia, the necessity of settling in Yukon has not yet arisen, but with the ever increasing demand for land and homes, the extensive and beautiful valley tracts of Southern Yukon must eventually be settled, and will undoubtedly some day support an agricultural population. Just how distant this time may be, it is now impossible to predict. In case, however, the mining industry should progress more rapidly, as is quite possible, a greater demand for agricultural products will arise, and the early settlement of the district will follow. Even under present conditions, the stock-raising industry presents a very attractive field for immediate enterprise. Every year many carloads of cattle are shipped into and through Yukon to supply the beef markets of this territory and Alaska. Also there is always a certain demand for horses within these territories. It would thus seem that the many miles of fertile valley tracts in various parts of Southern Yukon could be readily adapted to the raising of the stock at least to supply the local demand. Horses, it is known, can be easily and cheaply raised, and with an abundance of wild hay in places which could be put up for winter use, there would seem to be no reason why cattle raising could not also be made a success, particularly if hardy, northern breeds were selected. As with the agricultural products, however, stock-raising in the near future, will also be largely governed by the demand arising from the development of the mining industry.

FOREST[1]

Yukon as a whole is only sparsely forested, and nowhere do the dense growths of timber occur such as characterize portions of British Columbia and other localities to the south and south-

[1] Macoun, John, "The climate and flora of the Yukon district": Geol. Surv., Can., Ann. Rept., Vol. XV, 1902-3, pp. 38A-54A.

McConnell, R. G., "The Whitehorse Copper belt, Yukon Territory": Geol. Surv., Can., 1909, pp. 5, 6.

Cairnes, D. D., "Wheaton district, Yukon Territory": Geol. Surv., Can., Memoir No. 31, 1912, pp. 27-29.

"The Yukon-Alaska International Boundary, between Porcupine and Yukon rivers": Geol. Surv., Can., Memoir 67, 1914, pp. 10-12.

"Upper White River district, Yukon,": Geol. Surv., Can., Memoir 50, 1915, pp. 28-30.

east. However throughout all the southern portion of the territory, trees grow on mostly all the valley floors up to an elevation ranging in most places from 3,500 to 4,000 feet above sea-level, and on the mountain sides to practically the same height. Timber line is, however, characteristically not so high at the lower as at the upper ends of the valleys, and in places does not reach above 3,000 feet. In the valleys of Yukon, Lewes, White, Donjek, Teslin, Big Salmon and Little Salmon rivers and in the vicinity of Kluane, Sekulmun, Aishihik and other of the larger lakes, as well as at occasional other points, groves occur where the trees are tall and stand close together, but these are the exception. In general about one-third of southern Yukon is forested, the southern and western slopes being much better timbered than the northern and eastern hillsides. In this portion of the territory there is thus sufficient timber in most localities to meet the ordinary requirements of mining and allied industries for many years to come. In the immediate vicinity of Yukon and Lewes rivers, however, wood suitable for fuel is becoming somewhat scarce, due to the fact that the fuel used by the steamers plying between Whitehorse and Dawson, has, in the past, been almost exclusively wood obtained along the river banks. A relatively small amount of coal has, however, also been consumed.

As the Arctic ocean is approached, the forest growth becomes gradually less, so that by the time Porcupine river is reached, where crossed by the Yukon-Alaska boundary, at latitude 67°25′, only about one-fourth or less of the district is forested. The growth is there also noticeably more sparse than along the Yukon, and trees seldom occur at an elevation exceeding 2,000 feet above sea-level. Along the Arctic, very little timber of any kind occurs, the forest being represented by rare patches of small trees and shrubs along occasional stream courses.

The forest members of Yukon consists mainly of fourteen species that obtain the dimensions of trees. These include:—

White spruce.—*Picea canadensis* BSP.
Black spruce.—*Picea mariana* BSP.
Alpine fir.—*Abies lasiocarpa* Nutt.
Black pine.—*Pinus contorta*, Loudon.

Tamarack or American larch.—*Larix laricina.*
Balsam poplar.—*Populus balsamifera* L.
Western balsam poplar.—*Populus trichocarpa* T. & G.
Aspen poplar.—*Populus tremuloides* Michx.
Northern canoe birch.—*Betula resinifera* Regel.
or *B. alaskana* Sargt.

In addition a number of shrubs occur including:

Willow, several species.
Alder, two species.
Dwarf birch.—*Betula glandulosa* Michx.
Juniper.—*Juniperus communis* var. *montana* Ait.
Soapollali.—*Sherpherdia canadensis* Nutt.
Wild rose.—*Rosa acicularis* Lindl.

Of these, three species of willows and the two alders become in places small trees, and are thus included in the above mentioned fourteen tree species.

Of the different trees in the district, the white spruce is much the most useful and by far the most widely and extensively distributed, growing at all elevations up to timber line, and constituting in most places about one-half of the entire forest growth. The best groves are generally found on the river flats and in depressions along the lower slopes of ridges, where the trees are straight and well grown. Individuals are not generally more than 12 to 18 inches in diameter 3 feet from the ground, but specimens with 24 to 36-inch stumps were noted in a few places, but anything over 24 inches is exceptional. This tree furnishes a strong, easily worked timber, and is well suited to the usual needs of the miner, and to purposes of construction generally. Black spruce, alpine fir, and black pine, also grow somewhat plentiful in portions of Southern Yukon, but are not nearly so useful or abundant as the white spruce, and are mainly of importance as fuel. The poplars and canoe birch are also of considerable value as fuel in some localities.

TRANSPORTATION

Yukon reaches the coast only at the north where it is bounded by the Arctic ocean. The extreme southwestern corner of the territory, however, extends very close to the Pacific, being separated from this ocean by only a narrow fringe of land including portions of British Columbia and the "pan-handle" portion of Alaska. The most frequented route to Yukon is that via Skagway which is situated at the head of Lynn canal on the Pacific, 870 and 1,000 miles distant respectively from Vancouver and Seattle. From Skagway, the White Pass and Yukon railway passes over the Coast range of mountains, via the White pass, to the town of Whitehorse which is 110 miles distant from Skagway, and is situated at the head of navigation on Lewes river. Another easy, though longer route to Yukon, is that via Bering sea and Yukon river, navigation being open during the summer months from Whitehorse to St. Michael and Nome, two points situated on Norton sound near the mouth of Yukon river, 2,310 and 2,360 miles respectively from Seattle, and 1,600 and 1,700 miles respectively from Dawson, as measured along Yukon river. A number of other routes are also occasionally followed, the greater number of which head from Haines, Cordova, or Valdez,[1] points on the Pacific coast.

Within Yukon, all points in the vicinity of the White Pass and Yukon railway, as well as along Yukon river and its main tributaries and headwaters are thus readily accessible. Quite a number of roads and trails have also been constructed throughout the southern portion of the territory, which greatly facilitate access to certain localities. In addition, throughout Yukon, there are a number of long, prominent valleys which are more or less connected, and traverse the territory in different directions. These will afford excellent routes for railway lines when it is found advantageous to build such. A company has been organized for some years for the purpose of constructing a railroad from Haines on the Pacific coast, to Fairbanks on the Yukon, via Chilkat river, Dalton post, Lake Dezadeash, Lake Kluane, Kluane river, Koidern river, Beaver creek, Snag creek, Mirror

[1] Cairnes, D. D., "Upper White River district, Yukon": Geol. Surv., Can., Memoir 50, 1915, pp. 7-22; also see accompanying map 113A.

creek, and Tanana river. This route is quite feasible as far as location, grade, and ordinary problems of construction are concerned. A good grade and suitable location also exists from Whitehorse, the terminus of the White Pass and Yukon railway, westward to Lake Kluane. Also a series of connected valleys or depressions affords an excellent route from White Horse to White and Tanana rivers, via Ibex river, Takhini river, Hutshi lake, Aishihik lake, Nisling river, Wellesley lake, Beaver creek, Snag creek, and Mirror creek.

Thus when the necessity arises, there need be no lack of railway transportation facilities throughout Yukon, and particularly throughout the southern portion of the territory. At present, however, the building of these railroads would not seem to be warranted, but if instead, a number of good, trunk wagon roads were constructed, similar to the Whitehorse-Kluane road, which would crosscut and intersect the more promising portions of Yukon, and thus allow prospectors and others to prosecute their explorations and investigations with reasonable facility, such would be of great and immediate benefit to the territory.

Summary and Conclusions

Yukon is thus a vast territory concerning much the greater part of which very little or nothing is known. The prospected and explored areas, however, have proved to be very rich in natural resources. More than $157,000,000 in placer gold has already been produced, and there is estimated to be still between $30,000,000 and $45,000,000 remaining in the explored gravels of the few already producing districts. In addition the writer has estimated that within the explored portions of the territory there are probably 5,190,000,000 tons of coal in seams one foot or more in thickness, and this estimate is so conservative as to practically represent "known" instead of "probable" coal. Further, valuable lode deposits of economically important minerals are widely and extensively distributed throughout the known portions of the territory which undoubtedly constitute an important future asset to the territory. Also there are thousands of square miles of valuable valley lands which are well adapted to various

ranching and agricultural pursuits, and which, it is believed, will some day support an important agricultural population.

Thus with these and numerous other less important natural assets, the territory should have an important future and may be expected to play an important role in the development of Western Canada. Therefore, instead as has been suggested, of having reached an advanced stage in its economic development, due to the now rapidly approaching time when the Klondike placers shall have become exhausted, Yukon is as yet largely virgin territory, and should be considered as having only indicated by the slight development of the more or less prospected 17 per cent, what is yet possible from the remaining 83 per cent of its vast expanse.

THE WEEDON OR McDONALD COPPER MINE

By L. D. Adams, Weedon, Que.

Annual Meeting, Toronto, 1915

The property owned by the East Canada Smelting Co., Ltd., and known as the Weedon or McDonald mine, comprises lots 22a, b, c, and d, range 2, and lots 22a and b, range 3, of Weedon township, Wolf County, Province of Quebec, Canada. The active operations are being carried on by the Weedon Mining Co., Ltd., through a bond and lease given by the East Canada Smelting Co., Ltd. The total area is 375 acres, of which 150 have been cleared for mining purposes. The remaining area though poorly wooded will furnish sufficient timber for mining purposes for a number of years. The mine is 200 feet above the town of Weedon and 700 feet above sea level. The property is reached from Montreal by rail, either by the Grand Trunk or the Canadian Pacific systems, to Sherbrooke, thence by the Quebec Central Railway to Weedon, and thence by road, a distance of five miles, to the mine.

History

The property, though mentioned in the early geological reports of the Quebec government, was not actively prospected until 1908 and 1909. In 1908, John McDonald, attracted by the large area of gossan, succeeded in locating the ore-body in a large depression, after sinking many pits in the oxidized surface rocks. The first shaft was sunk in the centre of a large depression that conforms in size and shape to a horizontal section of the main ore-body. The outline of this depression is lenticular, having a major axis of about 570 feet and a minor axis of 60 feet. The No. 1 shaft, as the first shaft is called, was sunk vertically 19 feet, at which point the footwall was encountered, and thence onward for a distance of 80 feet it followed the footwall. At this

point the East Canada Smelting Co., Ltd., purchased the property and started development both by underground work and diamond drilling.

The production to date has been about 174,000 net tons having the following analysis:—Copper, 3.62%; Sulphur, 40.74%; Iron, 35.86%; Zinc, .77%; Lead, trace ; Insoluble, 12.2%; Aluminium, 4.08%; Gold, .01 oz.; Silver, 0.46 oz. The above tonnage sold for about $9.00 per ton, payment being made on sulphur and copper.

LABOUR

The labour problem has presented serious difficulties at times, since a large proportion of the employees are local French-Canadians, who cannot be depended upon to remain at mine work during the farming and timbering season. The average daily wage is $2.60. Contract work is encouraged in development work. The following prices prevail:—Raises, 6 x 10 @ $4.50 per foot; Drifts, 7 x 4 @ $6.00 per foot; Winzes, 10 x 5 @ $12.50 per foot; and Shafts, 16 x 6 @ $25.00 per foot. These prices cover all labour and supplies, except drill repairs, compressed air, and drill steel. In the cases of raising and drifting, the Company does the shovelling.

TOPOGRAPHY AND GEOLOGY

The property is situated on the side of a gentle ridge, that forms the eastern limit of the St. Francis valley. The apex of this ridge is a very acid granite, and the flank is composed of chlorite and sericite schists, the latter being the metamorphic equivalent of quartz porphyry. The ore has replaced a portion of the sericite schists, leaving chlorite schist as a footwall and the sericite schist as a hanging-wall. The main vein has been proved by drifting to have a length of 570 feet. There remains 10 feet of low grade copper ore, high in sulphur, still in the face of the south drift. The strike is N. 370° E. Sinking has proved a depth of 470 feet with no physical change along the dip, which is 45 degrees. The principal minerals are pyrite and chalcopyrite with an occasional stringer of galena and sphalerite. The non-metallic minerals are quartz, chlorite and sericite. The values in

the main vein are little affected by secondary enrichment from descending waters, no secondary minerals being found worthy of mention. Besides the main ore-body described there are two more veins of less importance. The No. 2 lies about ten feet into the footwall from the main vein and is parallel to it. The ore is low grade pyrite with low values in copper. The No. 3 is a pyrite vein 50 feet further into the footwall than the No. 2, and contains about the same values as the main vein, but is too narrow for successful mining.

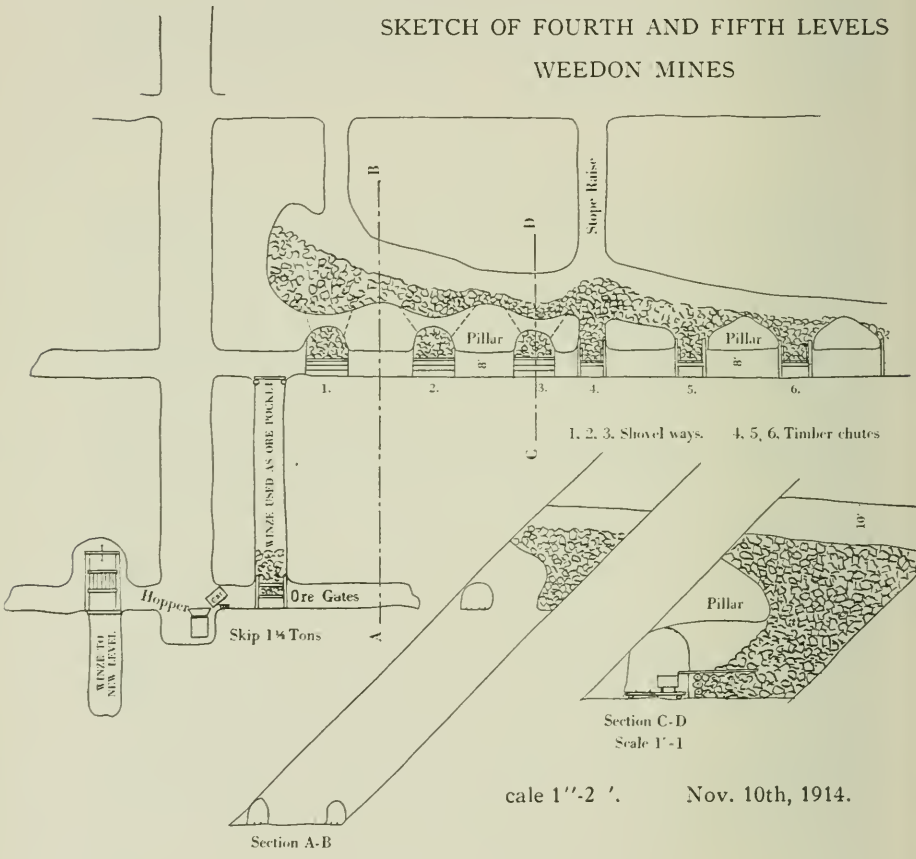
Surface Plant

The entire plant is being planned and equipped to use either electricity and compressed air or steam as power. The electrical equipment consists of the necessary transformers and controls for stepping the line current down from 15,000 volts to 2,200 for surface use and to 220 for underground use. When electricity is used for power a compressor of the latest belt-driven two-stage type is employed for driving the entire plant, with the exception of the pumps, which are direct-connected centrifugals having three stages and capable of handling 250 gallons per minute against a head of 400 feet. The compressor is driven by a 300 h.p. motor and compresses 1,500 cubic feet of free air per minute. The steam plant is composed of return tubular boilers having a combined capacity of 300 h.p. A straight line simple steam and compound air compressor having a capacity of 740 cubic feet of free air per minute furnishes air for drilling when steam is used as power. Two hoists are used, both having cylinders of 10 x 12 inches, and being of the geared type. The piping is arranged for either air or steam. All the necessary shops, sample mill and buildings have been erected.

Underground Workings

There are three shafts numbered in the order in which they were sunk. All three are sunk on the incline and follow the footwall, No. 1 shaft was the discovery shaft and no work was done in it below the first level. No. 2 is 350 feet deep and is situated

in the extreme north end of the main ore-body. This fact made it more economical to sink a new shaft situated in a more favourable position both for underground and surface work. To fulfil these conditions the No. 3 shaft was sunk and is now 450 feet deep. All depths are measured on the incline. There are six



levels averaging a little over 70 feet apart. The longest level is the second level, which is 570 feet long. The lower levels were all stopped when the copper content dropped below 2.5%; thus they are all about 400 feet long, with the exception of the sixth, which is not yet run out.

General Conditions

The following general conditions must be comprehended in order to fully understand the problems presented in mining the Weedon ore-bodies. First the ore is a compact mass of pyrite, having a specific gravity of 4.5. In practice we find 7.5 cubic feet in place to weigh 2,000 lbs. The ore drills easily, but is very hard to break. The angle of flow for general run-of-mine is about 70%. This, coupled with the great weight, makes handling very difficult. In the stopes the natural tendency is for the ore to break very large, and to pile up on the footwall unless the dip is very steep. Two methods of stoping are employed, the choice being governed by the angle of dip and width of ore. The mine makes about 200 gallons of water per minute. This water is very acid and contains so much fine sulphide slime that it is very hard on all pumps and pipe lines.

Development

The general scheme for opening new levels is as follows:—

A winze is sunk, 5 x 10, at a point about 12 feet from the No. 3 shaft on alternate sides. A portable head frame which can be lowered as each level is completed is used for this work. The hoist employed uses compressed air and has 5″ x 5″ cylinders and a gear ratio of 1.5. In sinking a contract is let to one driller, who drills, shoots, and furnishes his own powder, fuse and caps for $7.50 per foot. The shovelling is let on a separate contract. The shovellers receive $3.50 per shift for every shift that they completely clean the bottom; for all other shifts they receive $2.50. By reason of the nature and the weight of the ore, the speed of sinking is limited by the shovelling. The following programme has been generally followed:—The driller drills a centre cut employing six holes; he then drills one row about 1.5 feet behind each row of cut holes. This makes 12 holes, 4 feet long, drilled by one man with a small sinker per shift. He then loads and shoots. On the next shift the shovellers clean up what rock is broken. The driller then follows and plans his remaining holes, usually about ten holes, 4 feet long, from the results obtained in shooting the cut. In this way no time is lost from the failure

of any of the cut holes, the round is certain to be squared up every fourth shift and the shovellers are given all the rock they can possibly handle. About 50 feet is sunk per month at a cost of $14.50 per foot. As soon as the next lowest level is reached a drift is run under the shaft and the station and the sump is cut. This is usually done by drifting across the station on the foot-wall, then side-slabbing until the hanging wall is reached. The ore broken in station work is counted as stope ore and costs, placed in the skip of the level above, about $1.20 per ton. As soon as the station is completed a raise is made, the size of the shaft (16 x 6 in the clear), to connect with the shaft above. The timbers and track are then brought to the new level and all loading from the level above is stopped. This is accomplished by putting a chute in the bottom of the winze and using the winze as an ore pocket. A grizzly is placed over the top to screen out the large boulders coming from the stopes. The boulders are then block-holed and dropped into the pocket. A car is designed to hold just one skip full of ore and the skip tender is able to load this car and push it in position for loading the skip in less time than it takes for a skip to be loaded, hoisted, dumped and returned. The skip holds one and one half tons, and at a maximum, twenty skips can be hoisted per hour from the bottom level. The advantages of this method of opening a new level as against sinking the main shaft are as follows:—The water that finds its way into the shaft does not have to be pumped during sinking. Only about half the amount of rock is handled with a shovel as would be by sinking the main shaft. An ore pocket is cut without cutting any waste. Hoisting is not interrupted in the main shaft.

DEVELOPING THE NEW LEVEL

As soon as the skip is able to operate from the new level, drifts are started both ways from the shaft on both the hanging and foot walls. These drifts are run until the distance between the two walls is less than fifteen feet, then the hanging-wall drift is turned towards the foot-wall and carried ahead on the foot-wall until the limit of commercial ore is reached. The next step is to crosscut between the two drifts every twenty feet. These crosscuts are carried about 12 feet wide and 5 feet higher than the tops of the wall drifts. Broken ore is allowed to pile

up in the crosscut until a 'solar' can be laided on the broken ore at the right height to form a shovel-way and lip for loading the cars. All drifts are carried high enough so that the cut holes can be pitched very steep. This is necessary, because of the difficulty in getting the cut holes to break. In drifting, as in sinking, provision is made for squaring up the round by a second set of holes planned after observing the work done by the cut holes. The drilling, powder, fuse and caps are furnished by the contractor for $6.00 per foot. The shovelling is done on Company account. An effort is made to always have a clean drift ready for the contractors. It is very rare that a driller is compelled to touch a shovel and thus he is able to devote his entire time to drilling. The usual programme in drifting is as follows. The contractor drills his cut holes, three in number, about five feet deep and having a dip of about 70 degrees. From the same position of the arm he drills three holes, one above each cut hole. Each of these is drilled 4 feet deep. Then moving his arm, he drills two and sometimes three lifters. He then shoots, using 40% Forcite powder. On the following shift, the shovellers clean out the broken ore and bring in the drill and tools ready for the next set up. The driller then follows and drills the necessary holes for squaring the round, usually about nine holes, and shoots. Thus a drift round is squared every four shifts as in sinking. It is admitted that the methods here outlined are slow, but the advantages gained more than balance the loss in speed. Compared with results from a system designed for speed in driving, the cost per foot has been lowered from $16.00 per foot to $10.00 while the average distance driven per month has been reduced about 10 feet only per heading. To fully appreciate this method one must remember that the limiting point in all development work at Weedon is the shovelling due to the great weight of the ore. It has been found that the programme outlined makes all the ore that can possibly be handled by the shovellers without leaving some to be shovelled back by the drillers, which would add to the cost, as shovellers receive only about 80% the wage of drillers. Raises are driven from the level below to the level above, following the foot-wall, every hundred feet along the strike. These raises serve for ventilation, pipeways, and manways when stoping is started.

STOPING

At this stage in the development stoping is started. The plan of attack depends on the dip and width of ore. In the narrow portion of the vein raises are made every twenty feet along the strike. When these raises have reached a height of fifteen feet, timber chutes are placed in each and a sub-level is driven at the top connecting them, each with the other, and forming the bottom of the new stope. The stope is then carried up as an ordinary shrinkage stope, starting from one of the permanent raises as all the chute raises fill with broken ore. The air lines are brought down from the level above through the same raise. In the Weedon ore the increase in volume is about 40 per cent; hence 40 per cent of the ore broken can be drawn at once. Canadian Rand butterfly valve stope drills are used. Each driller drills 40 feet of hole per shift and breaks 30 tons; thus every foot of hole breaks .75 tons. Stope drillers receive $2.50 per shift. The costs for drilling, including block-holing in the stopes, is $0.13 per ton. The cost for powder is $0.13 per ton. The repairs on a stope drill average $0.25 per shift or less than $0.01 per ton broken. In the wider portion of the vein or where both foot and hanging-wall drifts are possible, the foot-wall drift becomes the bottom of the stope. In this case the back of the foot-wall drift is carried up high enough to give sufficient thickness to the pillars left over the shovelways to support the broken ore. If the dip is 45 per cent or greater the stope is then widened until all ore from wall to wall is included. The broken ore is drawn through the shovelways described above, and the stope is then carried up as in the narrower portions. Where the dip is less than 45° the foot-wall back is carried up about 8 feet wide until half of the stope area has been mined. The broken ore is then drawn out and by drilling long holes almost normal to the dip the remaining ore is broken to the hanging-wall. When this operation has reached the same height as the foot-wall cut, the cut is then carried to the level above and the same operation is repeated. The shovel always shows a much lower cost than the timber chutes. This is due to the fact that very little timbering is required for installation. Again in blasting boulders, the shovelways allow boulders to be blasted at the end

of each shift, whereas in the timber chutes, boulders must be blasted very frequently as they block the chutes. The cost of repair is high in the timber chutes as the blasting soon destroys the lagging. Trammers working from the timber chutes average about 30 one-ton cars per shift for two men, while from the shovelways 40 one-and-one-half ton cars are taken. Before stoping is started all tracks are removed, as the ore falling on the rails during development soon destroys both rails and grade. The permanent 20-lb. rails are laid on ties which take both tracks. The ties are laid on 6-inch poles thus leaving a good drain under track. When laid in this way, the tracks rarely need any repair, and outlast the stope they serve.

The ore is hauled from the stopes and development work in cars of 18 cubic feet capacity, which are found to contain 2,200 lbs. Two men are employed on each car. The average number of cars per shift, when the shovelling is from rock bottoms, is 20 cars, from shovelways 40, and from chutes 30. The ore is trammed by hand over an average distance of 100 feet to the ore pocket, where it is dumped. Hoisting is done in a skip which holds 3,000 pounds. The hoisting of ore is concentrated into about four hours per shift during which time 100 tons is hoisted, or at the average rate of 200 tons, per 24 hours. This quantity can be increased to 400 tons per day, but that would necessitate crowding the trawmay and cause excessive wear on the cables. The skip dumps into an ore pocket of 400 tons capacity. The ore is drawn from this pocket through four gates of a design developed at Weedon to overcome the destruction, consequent on the great weight of the ore and the abrasive effect caused by the high sulphur content, of gates of the common types. The ore from the gates flows directly into the tramway buckets which circle around the bin thus allowing ore to be drawn from any side of the bin. The bin is flat bottomed and by careful work the wear on the bottom and the sides can be minimized, as a pyramid forms in the centre of fine ore which receives the impact as the ore fails from the skip into the bottom. The tramway buckets hold 850 lbs.; and one bucket goes out every minute. Where the tramway reaches the railroad, the ore is dumped into bins holding 2,000 tons. Chutes are arranged for loading open cars in summer and box cars in winter. In loading box-cars the

ore is drawn through chutes into wheel-barrows and wheeled into the cars. In this way the ore is trimmed without shovelling. The cost for loading open cars is .04 cents per ton while box-car loading averages .07 cent.

The head-frame is designed and equipped for crushing all ore to 4 inches and for sorting, but thus far there has been no need of either.

SAMPLING

Cut samples are taken every ten feet during development and plotted on the assay map. Grab samples are taken after each blast and plotted on a drift record which is kept on each heading being worked. Car samples are taken to check the stopes in case the value of any stope is in doubt.

MAPS

A stope map is kept showing the monthly progress in all work. This map is plotted on a plane parallel to the dip, thus all distances are true distances. A map of all the work is brought up to date each month.

COSTS

Detailed costs will be given for an average month. This includes all charges except superintendent, manager, shipping office at Portland, Maine, where the ore is transferred from the railroad to barges, insurance taxes and home office.

During the month in question 5,400 tons were shipped.

MINING COSTS FOR OCT., 1914

	Total	Per Ton
Supplies	$ 3,086.00	$ .57
Labour	2,099.61	.39
Total	$ 5,185.61	$ .96

DEVELOPMENT COSTS, OCT., 1914

	Total	Per Ton
Labour	$ 1,694.17	$ .313
Supplies	451.65	.083
Total	$ 2,145.82	$ .396
General	900.00	.168
Repairs and renewals	1,009.52	.187
Car loading	423.42	.078
Tram operation	362.62	.067
Total	$ 4,841.38	$ .896

That the above costs may be adequately appreciated, a few unit costs will be added. Each stope driller breaks 30 tons per shift. Each shoveller handles 11 tons from (rock) bottom and 22 tons from shovel-ways per shift. Each man underground produces 5 tons per shift. Each man loading, loads 27 tons per shift. Each man on the payroll, including office help, produces 2.7 tons per shift. Each piston machine drills 40 feet of holes per shift, and, in development work, each foot of hole breaks .40 of a ton. Each hammer drills averages from 40-45 feet of holes per shift and each foot of hole breaks .75 tons. Each ton shipped requires 1.65 h.p. per 24 hours.

DETAILED COSTS

	Labour Mining		Labour Development	
Hoisting engineers	$215.00	.039	$ 71.68	.014
Firemen	23.48	.004	7.82	.001
Blacksmiths	231.26	.042	77.09	.015
Drilling	732.25	.135	1,069.48	.188
Shovelling	546.36	.10	182.11	.034
Timbering	65.25	.01	62.75	.012
Pumping	115.32	.02	38.44	.007
Mine office	180.00	.03	.....	....
Miscellaneous	611.31	.113	184.80	.034
Boarding House def	195.06	.036	.....	...
Total	$2,915.29	.529	$1,694.17	.305

	Supplies Mining		Supplies Development	
Fuel Tramway	$ 5.93	.001	.....	....
Fuel	235.25	.044	.....	....
Lubricants	52.75	.009	.....	....
Waste and packing	81.32	.001	.....	....
Powder	622.00	.119	397.48	.073
Fuse	77.28	.015	48.75	.008
Caps	25.18	.004	10.42	.002
Lights	114.27	.02	.....	....
Lumber	39.90	.007	.....	....
Assay	58.29	.011	.....	....
Power	826.04	.153	.....	....
Miscellaneous	33.94	.006	.....	....
Total	$2,172.15	.39	$456.65	.083

	Repairs	
Engines	$ 18.12	.003
Boilers	24.13	.004
Hoists	65.41	.012
Compressor	21.25	.003
Drills	251.38	.046
Pumps	149.88	.028
Blacksmith shop	26.50	.004
Pipe lines	174.59	.032
Cars	51.31	.009
Ore Bins	61.00	.012
Office building	43.71	.008
Machinery	61.96	.012
Engineering office	48.68	.009
Sampling mill	440.65	.007
Motors	25.75	.004
Total repairs	$1,464.32	.193

THE OXYGEN IRON TORCH

By David H. Browne, Metallurgical Engineer, The International Nickel Company, New York.

Annual Meeting, Toronto, 1915.

One of the most disagreeable features of blast furnace and reverberatory work is the amount of 'muscular metallurgy' which, in spite of our progress in other directions, still remains with us. Metallurgy seems to entail, as ever, much sweating and profanity. Every step we make toward the elimination of hard work renders it easier to retain valuable employees, whose value is more a matter of their brains than of their muscle.

The operation of tapping a blast furnace settler or a reverberatory furnace is sometimes very difficult. Conditions sometimes occur where the matte chills, or the tapping bar breaks in the hole. This necessitates hard sledging for hours and leaves the workmen physically exhausted.

To avoid this, use has sometimes been made of the electric arc. Grounding one wire on the iron work of the settler or furnace and using an electric carbon at the end of the other wire is sometimes very successful. This scheme was used by the Carnegie Steel Company's Armor Plate Department, at Munhall, and from their description was tried at the Canadian Copper Company's furnaces at Copper Cliff, Ontario.

The Superintendent of the Armor Plate Department, Mr. S. S. Wales, gave me in 1905 the following description of this apparatus:—

"We have used 250 volts direct current for this purpose, in which case we grounded one end of the line and attached the other end to the carbon, to be used to burn with, which is carried on a long wooden handle. About 300 to 500 amperes are used and the voltage reduced by use of resistance, the power consumed being about 125 H.P. The carbon used would be about 2" to 3" diameter by 6" to 12" long.

"The best method of accomplishing the results with alternating current is to use a single phase transformer of about 50 K.W. capacity, mounted on a four-wheel truck, so that it may be pulled to the place where you wish to use it. The primary can be connected to one phase of your circuit, if you have 3 or 2 phase current, or across the line if you use single phase. It would be more convenient to use a higher voltage than 500 volts for the primary connection as the size leads would be somewhat reduced. However, 500 volts a.c. will answer the purpose, and the secondary coils of the transformer should have taps brought out at for 50, 100 and 200 volts. Generally you would use the 100 volt tap and a current of from 800 to 1000 amperes in the secondary. Such transformers are constructed by the Westinghouse Electric Company

with a view to maintaining a steady current under the difficult conditions which are met with in this work.

"You will find several advantages in the use of this outfit. In the first place it does not ground the main line or generator, since the second is, of course, entirely isolated from the leader system. In the second place, the amount of power consumed with a given current in the arc is far less than with direct current, since there is no loss in the rheostat. In the third place, the current in the arc is far more steady and more easily controlled, and therefore more effective. It generally takes about one half hour to burn out the tapping hole, though when starting a furnace after a long shut down may it take longer. It is generally better to cut out the slag and clay with bars, and then to burn, for say 15 minutes, after which the bars should be used again, and so on until the hole is open.

"Care must be taken in using any apparatus where a large electric arc is used, not to expose the face or eyes, unprotected, to its direct rays, as a severe 'sun burn' will result, which will cause the skin to peel off and the eyes to be severely inflamed even though no heat or burning may be noticed while using the apparatus."

As will be seen from this letter, the operation, while more rapid than sledging, is by no means instantaneous. In crowded quarters there is great danger of accident from short circuiting. Furthermore, it requires considerable time to set up the apparatus. For this reason the electric arc has but a limited application as a means of cutting out a frozen tap hole.

In September, 1913, while I was visiting the plant of the Sulitjelma Copper Company, at Sulitjelma, in Norway, I was greatly interested in a device used with great success by Mr. Westly, the Superintendent. This consisted of an oxygen iron flame, introduced into the tap hole. The speed with which this acted was amazing—work that would require several hours' sledging being done in less than a minute.

As used by Mr. Westly, this consisted of a movable oxygen tank containing the gas under considerable pressure. This tank was carried over to the furnace and connected by a piece of rubber tubing to a piece of small iron pipe usually about one eighth of an inch in diameter, and as long as could be conveniently handled. The end of the pipe was heated white hot by inserting it into the slag stream and the gas was turned on slowly until the iron pipe was sparking in brisk combustion. The pipe was then thrust into the tap hole and the gas turned on more fully. An intense heat was immediately generated. The pipe was pushed close up against the obstruction, which immediately melted, all the slag oxides and molten particles being blown backward out of the tap hole by the escaping gases.

This novelty was of such practical interest that on my return to Copper Cliff, Mr. Kent, the Superintendent of the smelter, put the idea into use with immediate success, and for the Canadian Copper Company the days of hard sledging were past history.

In June, 1914, I brought the matter to the attention of Mr. Mathewson, of the Anaconda Copper Company. A letter from Mr. Carlyle of that company, dated July 27th, 1914, shows the immediate application of this device to their own conditions. Mr. Carlyle says:

"With regard to the oxygen torch, we used oxygen compressed to 300 lbs. The tank was taken up to the converter or reverberatory taphole as required and connected by means of a rubber hose to a length of $\frac{1}{8}$" pipe. The gas was turned on gently and the end of the pipe ignited either in the furnace or by means of a small fire made with two or three pieces of coke. As soon as the iron was giving off sparks, as iron does when at the welding heat, we would thrust it into the tuyere or taphole and increase the flow of oxygen. We were able to burn out several inches of copper in a converter tuyere in a few seconds, and a reverberatory taphole, on which the foreman had had his whole force working with sledges for two or three hours, was burned out in under 2 minutes.

"The Company is putting in the necessary compressors to tank their gas under 1800 lbs. pressure. The tanks used will be much smaller than the 300 lbs. tanks, and it will be very simple to design a small truck equipped with one of these tanks and the necessary hose and iron pipe. This apparatus can be wheeled up to whatever point it is needed.

"We also used the torch in some experiments in burning out slagged tuyeres from the blast furnace. I think this would be quite feasible if done with such care that the thimble was not injured, but we did not carry the experiments very far."

In practice care must be taken to provide a sufficiently long piece of pipe, as it is rapidly consumed by the oxygen. In order to protect the workmen from molten slag and oxide blown out behind the pipe, a shovel with a notch or hole in it through which the pipe passes is laid over the tap hole.

The entire operation seldom takes over a minute from the time oxygen is turned on. The gas should be at least at 300 lbs. pressure and the higher the pressure the better.

It is quite possible to use this apparatus to cut apart structural wreckage, such as heavy beams or girders, where haste is an important matter.

The oxyhydrogen or oxyacetylene flame, has perhaps more applications as a constructive tool, but as a means of quick and satisfactory destruction the oxygen torch occupies a field that is all its own.

RECOVERY OF MERCURY FROM RESIDUES OF AMALGAMATED COBALT ORES

By E. B. Thornhill., Cobalt, Ont.

Annual Meeting, Toronto, 1915

In submitting this paper on the recovery of mercury as sulphide from the residues from the amalgamation and cyanide treatment of high grade cobalt ores and concentrates, I do not care to enter into a discussion of the many reactions, chemical and otherwise, that take place in the general process, but wish to confine myself more particularly to the recovery of mercury from the residues.

I submit charts of two amalgamation-barrel charges, which will be attached as illustrations to this paper, and are self-explanatory.

In the amalgamation of Cobalt high grade silver ores and concentrates in strong cyanide solution, as practised in that district, considerable mercury is contained in the residues from the amalgamation process. This mercury exists in the residues as mercuric sulphide in most part, only from five to ten per cent of the total mercury content being in the metallic state. Attempts to eliminate this loss in the amalgamation process were tried, but these all resulted in a low extraction of the silver content of the ore, and attention was then directed toward the recovery of the mercury from the residues.

The process developed at the Buffalo Mines for this purpose consists, in short, in leaching out the mercuric sulphide with a caustic, alkaline sulphide solution, and precipitating the mercury from solution with metallic aluminum. The equations for solution (1) and precipitation (2) are:—

(1) $Hg\,S + Na_2S = Hg\,S \,.\, Na_2S.$

(2) $3\,Hg\,S \,.\, Na_2S + 8\,Na\,OH + 2\,A_1 = 3\,Hg + 6\,Na_2S + 2\,Na\,A_1\,O_2 + 4H_2O.$

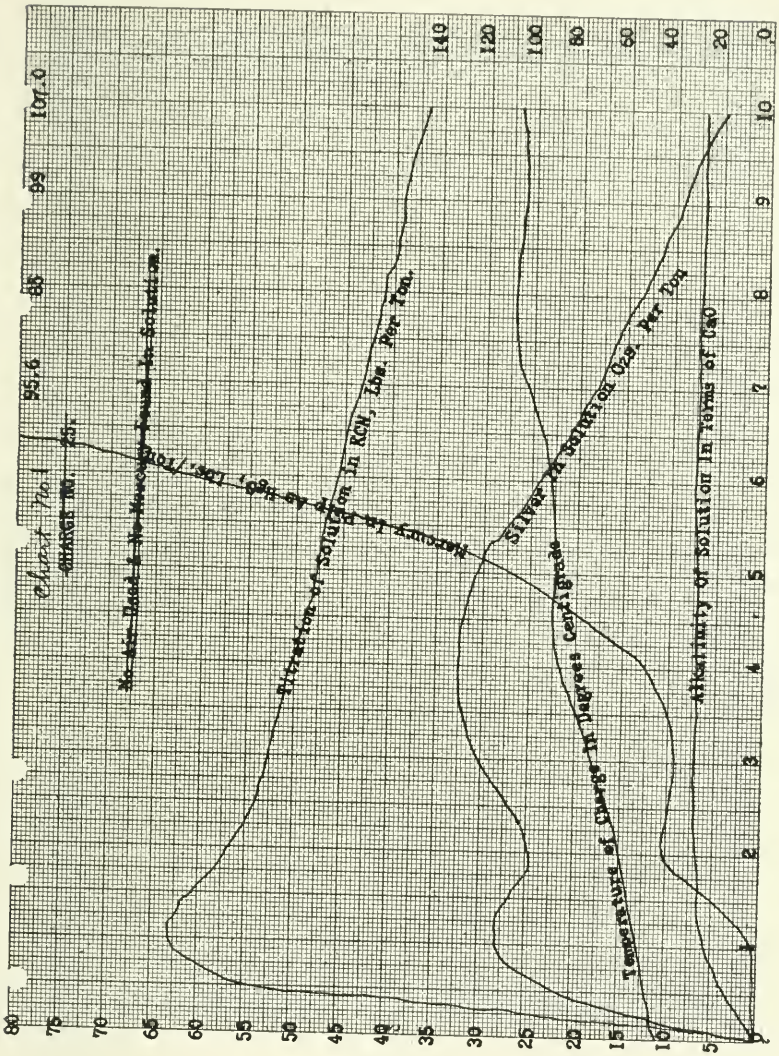


Chart No. 1
Lbs. Hg. Recovered to Solution
Fraction of Solution in Ref. Lbs. Per Ton.
Silver in Solution Ozs. Per Ton
Percentage of Mercury in Original Concentrate
Affinity of Solution in Terms of CaO

Small scale experiments showed that a complete extraction of the mercuric sulphide could be made by an eight to ten-minute treatment of the residues with the alkaline sulphide solution.

Advantage was taken of this fact in the commercial plant by applying the solvent to the residue on the filter leaf, as no agitation of any sort was required.

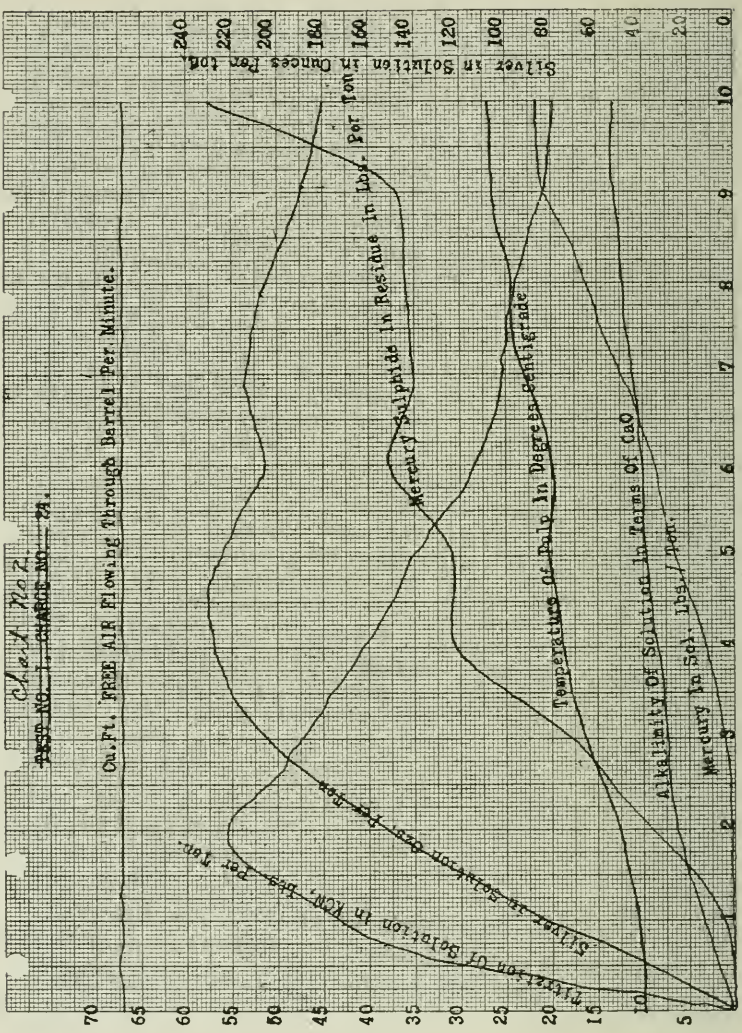

Cu. Ft. FREE AIR Flowing Through Barrel Per Minute.
Silver in Solution in Ounces Per Ton.
Mercury Sulphide in Residue in Lbs. Per Ton.
Temperature of Pulp in Degrees Centigrade.
Alkalinity Of Solution in Terms of CaO
Mercury in Sol. Lbs. / Ton.
Time in hours.

Chart No 2

The operation of the constructed plant is essentially as follows: The residue in the pregnant cyanide solution is caked on a Moore filter leaf of the usual construction and the cake washed free of silver solution with water. The basket is then lowered into the sodium sulphide solution and this solution drawn through the cake until the effluent shows only a trace of

mercury. Usually one ton of solution per ton of residue is sufficient. This mercuric sulphide solution is pumped to a precipitating tank and the mercury precipitated by adding granular aluminum to the agitated solution. Agitation is then stopped, the precipitate allowed to settle, and the clear solution decanted. The precipitate is then run to a small wash tank by sweeping it out through a hole in the bottom, employing a raking mechanism similar to a Dorr thickener, having pieces of old rubber belting riveted to the bottoms of the rakes. The precipitate is then washed with water by decantation and drawn off into a steam drying pan. After drying, the fluid mercury is separated from the powdered metallic by raking the latter off with a hoe. The fluid is strained through canvas and is ready to return to the circuit. The powdered material containing approximately seventy-five per cent mercury is then retorted and the mercury condensed in the usual manner.

The strength of the solvent used is kept up to four per cent sodium sulphide and one per cent sodium hydroxide. Weaker solutions can be used with equally good results but the quantity of solvent solution required was found to be directly proportional to the strength; that is with a four per cent sodium sulphide solution, one ton of solution would extract the mercuric sulphide from one ton of residue, but if a two per cent solution was used, two tons would be required per ton of residue treated. The concentrated solution offers the advantage of less solution to handle and an economy of aluminum in the precipitation. The sodium sulphide used is the commercial salt, costing $1.25 per cwt. in barrels f.o.b. cars Toronto.

The precipitant used is a waste product of aluminum casting foundries, containing seventy-five per cent aluminum, the impurities consisting of varying percentages of copper, silica, wood, waste, grease, etc., the grease being removed before using the aluminum by burning it off. About one-quarter pound of this material is used per pound of mercury precipitated.

It was anticipated that the sodium aluminate would accumulate in the solution to such an extent that some special means would have to be taken to remove it. This, however, has not been the case. Some aluminum as the hydrate, falls with the mercury on precipitation, and some is removed during the process

of leaching, presumably as calcium aluminate, which collects on the filter cake.

The precipitation of the aluminum out of the solution by these means regenerates caustic soda so that the consumption of this chemical is reduced to one-tenth of a pound per pound of mercury recovered, instead of one-half pound as shown by the theoretical equation for precipitation. Sodium sulphide is also regenerated in precipitation but there is also a mechanical loss of approximately twenty per cent of the solution used in leaching, as no water wash is used to recover the retained sulphide.

From May, 1914 to date, 37,650 lbs. of mercury have been recovered at a cost of approximately 13c. per pound for labour and chemicals.

The laboratory methods of determining the mercury in the different products of the process may be of interest. To determine the total mercury (both metallic and sulphide) one-half gram of the material is well mixed with cast iron filings, free of grease, and the mixture placed in a hard glass tube, sealed at one end with a contraction about two inches from the sealed end. The mercury is distilled off by heating the bulb containing the charge, and condenses in the tube just beyond the contraction. After the distillation is complete, the contracted portion of the tube is heated up, the bulb portion pulled off and the end of the tube sealed. The tube containing the condensed mercury is allowed to cool, filled one-half full with a one-tenth per cent KCN solution and ten to fifteen 100 mg. gold beads added. Each bead will amalgamate with about one mg. of mercury. Shake until all the mercury is amalgamated, transfer beads to a small porcelain cup, wash with water, dry with alcohol and weigh. Retort off the mercury and weigh again. From ten to twelve determinations can be made in one hour by this method and it is accurate enough for control of operations. When it is necessary to know the proportionate amounts of mercury as metallic and as sulphide, two 2-gram samples are weighed out, one digested with concentrated nitric acid and the other with a ten per cent solution of sodium sulphide and the mercury in the residue from each determined by the method just given. The strength of the sodium sulphide solution is determined by

titrating against a standard zinc chloride solution using sodium nitro prusside as an outside indicator.

The mercury produced by this process is of exceptional purity. Ledoux & Co. report one-quarter oz. of silver, it being the only impurity in determinable quantity. This mercury is at present being used by the leading electrical concern in the United States for its mercury convertors, without purification.

In conclusion, I wish to acknowledge my indebtedness to Mr. T. R. Jones, General Manager of the Buffalo Mines, for his practical suggestions for economy in the process.

Discussion

Mr. Arthur A. Cole.—Anyone familiar with the practice in Cobalt knows that when the high grade plants were started it was hoped that a process had been devised that would compete with the smelters; and one of the difficulties experienced was that a certain proportion of mercury went into the residues. In the Nipissing they overcame this difficulty to a certain extent, because they used only raw ore; when they used table concentrates the mercury ran up much higher. As at the Buffalo the cyanide plant was not as complete as that at the Nipissing, it was necessary to discover some means of extracting the mercury from the residues. The subject was attacked by the management, and Mr. Thornhill commenced experiments which he brought to a successful issue. The plant has now been in operation for some months, and instead of buying mercury, they now have mercury for sale, as they have more than they can use. They have worked over the old dumps of residues, and have recovered most of the mercury lost. A very interesting point is the fact that the mercury recovered is such pure mercury. The difficulty with the commercial mercury is that it almost always contains a certain amount of arsenic, which in some electrical appliances is deposited as a thin film, interfering with the efficiency of the contact. The mercury collected by the new process is absolutely free from arsenic, although it had been associated in the process with some of the most arsenical ores in the world.

METALLURGICAL PRACTICE IN THE PORCUPINE DISTRICT.*

By Noel Cunningham, Douglas, Ariz.

Many excellent descriptions of the mills of the Porcupine district have been written, but no discussion exclusively devoted to the metallurgical technology has been given. These notes are intended to cover this feature briefly. They are based upon 2½ years' mill operation in the district—i.e., practically since the beginning of metallurgical operations.

Character of the Porcupine Ore.—There is no oxidized ore in the district, the surface having been deeply planed by glacial action in recent geologic time. The precious-metal content is about in the proportion of 85 of gold to 15 of silver by weight; hence, the silver is practically negligible. There are two classes of Porcupine ore, having very different characteristics; these will be referreed to throughout this paper as Class A and Class B.

Class A ore is a pure quartz with inclusions of schist. Generally it is heavily fractured and breaks down readily to sharp, hard grains about minus 10 plus 20 mesh, requiring further comminution to release the gold. It carries very little pyrite; the gold is entirely free and apt to be coarse, but often spongy, going into solution readily on that account. This gold is 60 per cent to 85 per cent free milling, depending on the grade of ore.

Class B ore is an iron silicate schist, strongly laminated, carrying 4 to 5 per cent pyrite; its specific gravity is 2.8 to 3.0, depending upon the amount of mineralization. In breaking the ore in the mine, generally over 25 per cent of material through ½ inch ring is made; the ore readily breaks down in milling and makes a comparatively large amount of non-crystalline slime; owing to its high specific gravity, however, it is quick settling. In my opinion, the gold in this ore is free, but so finely divided that it will neither pan nor amalgamate; it appears to

*Presented also, by joint agreement, before the American Institute of Mining Engineers.

be disseminated through the rock and not chiefly associated with the pyrite.

Veins of Class A ore occur with or without side walls of Class B, and veins of Class B occur unassociated with Class A; more often the veins are closely banded, Class A and Class B alternating, generally with Class B in excess. Both classes of ore are more or less blocky at times, and with reference to Class B this is indicative or low gold content.

From a treatment standpoint neither class of ore introduces any important difficulty, although there seems to be a tendency toward reprecipatition, due probably to some element in Class B material. Practically no cyanicides are present in the ore, chemical consumption being about 0.2 lb. of cyanide per ton of ore; 1-lb. cyanide solution is sufficient for extraction, and protective alkalinity may be carried very low. With a well-designed battery and tube mill installation, a stamp duty of 15 tons or better can be readily maintained.

OUTLINE OF TREATMENT AND DEVELOPMENT AT THE PRINCIPAL MILLS

Although excellent descriptions of the mills and treatment methods of the Porcupine district have appeared in the technical press, it will be of benefit to outline the treatment in the five principal mills of the district in chronological order, and to comment briefly on the metallurgical trend indicated.

First Hollinger Mill.—Destroyed by fire before ready to operate. Treatment intended: Fine crushing, plate amalgamation, and concentration of tailing.

First McIntyre Mill.—Designed for treating Class A ore. Crushing by 10 light stamps, fine grinding, plate amalgamation, and concentration of tailing from amalgamation. As the mine developed, chiefly Class B ore was produced, from which an extraction could not be made by amalgamation. This mill was shut down after about a year's run.

Vipond.—Mill of 100 tons capacity treating a mixture of Class A and Class B ore. Treatment: Fine grinding and plate amalgamation. Simple amalgamation did not make a satisfactory recovery of the gold and the mill was shut down after a few

months' run. Recently the mill resumed operation, amalgamation having been abandoned and a cyanide plant added. Treatment: Fine grinding in cyanide solution, agitation, and complete counter-current decantation.

Dome.—A 40-stamp mill, recently increased to 80 stamps, treating a mixture of Class A with a less amount of Class B ore. Treatment, *at start:* Stamping in water, primary amalgamation, fine grinding, secondary amalgamation, dewatering, agitation in cyanide solution, Merrill filters, to waste. *Later:* Stamping, tube milling, and plate amalgamation in water, cone classification, to three products; (a) slime, dewatered and agitated in cyanide solution, Merrill filters to waste; (b) sand, leached; (c) concentrate, reground in tube mill in closed circuit with classifier and amalagmation plate, classifier overflow to slime treatment.

Hollinger.—A 40-stamp mill, recently increased to 60 stamps, treating a mixture of Class A and Class B ore, the latter predominating. Treatment, *at start:* Stamping in solution, fine grinding, concentration, concentrates amalgamation in solution and returned to table tails, table tails to agitators, to Moore filter, to waste. *Later:* Stamping in solution, fine grinding and concentration, concentrates agitated in strong solution, washed and impounded; table tails to two steps of continuous decantation, to filters, to waste. *Now building:* Plant for agitation and complete counter-current decantation for one-third of the tailing from table concentration.

Porcupine Crown.—Plant with 10 light stamps later increased to 20, treating Class A ore entirely. Treatment, *at start:* Stamping and fine grinding in water, followed by plate amalgamation. *Later:* Stamping and fine grinding in solution, with plate amalgamation in closed circuit with tube mill and classifier, followed by agitation and complete counter-current decantation.

Second McIntyre Mill.—Plant of 150 tons capacity recently increased to 300 tons, treating a mixture of Class A, with large preponderance of Class B ore. Treatment, *at start:* Fine grinding in solution, agitation, Burt filter to waste. *Later:* The capacity was increased from 150 to 300 tons, the treatment being

unchanged except that continuous decantation replaced filtration in the new unit.

Acme Mill.—Now building with 40 stamps, to treat a mixture of Class A and Class B ore, the latter preponderating. Treatment: Stamping and fine grinding in solution, agitation, concentration, concentrates to be reground in solution, agitated, washed and impounded, table tails to be treated by decantation.

ANALYSIS OF THE MILLING PRACTICE

A tendency toward extensive alteration in treatment methods will be at once apparent from a consideration of the above outline of milling practice and development. This is chiefly due to the fact that large bodies of Class B ore are now being developed and treated, whereas the design of most of the mills was determined almost entirely from tests upon Class A ore. The entire failure of straight amalgamation is obvious. Amalgamation in conjunction with cyanidation is practiced only at the Dome, where large bodies of Class A ore are yet to be treated, and at the Porcupine Crown, where to date only Class A ore has been found. At the latter mill, however, only a small plate area is used in the classifier-tube mill closed circuit, as the ore contains a large amount of coarse free gold which is readily caught at this point.

The equipment of the Porcupine mills offers good opportunities for comparison of various machines for doing the same work.

Stamps versus Rolls and Hardinge Ball Mills.—At the Vipond and the McIntyre mills, rolls and ball mills are doing the work done by stamps at the other mills. The ore is chiefly soft schist and the ball mills have been entirely satisfactory; power per ton of ore ground appears to be slightly higher than with stamps for the production of identical results. Steel consumption is about the same, the stamps perhaps having a shade the better of the argument in this respect; cost of operation and repairs is in favor of the ball mill, while first cost and uniformity of operation (what might be termed lack of operating "grief") are decidedly in favor of the ball mill. While my own experience in this district has been entirely with stamps, and their

performance was satisfactory, I am of the opinion that the ball mill is preferable for breaking down the Porcupine ore ahead of the tube mills.

It may be of interest to note in passing that at the McIntyre a first-class Chilean mill was discarded in favour of the ball mill after the two had run side by side for a year. At the Vipond, Hardinge pebble mills are used for fine grinding, while cylindrical mills are in use in all the other mills. I do not know how the conical mills compare with the cylindrical mills in first cost, power required, performance, etc., but on theoretical lines I favour the cylindrical mill, where coarse gold is to be dissolved in solution, as more effectively trapping the gold particles and wearing them down to microscopic fineness, owing to the vertical end of the mill.

Stamping in Water and Amalgamating versus Stamping in Solution with No Amalgamation.—Probably the best opportunity for studying this point is afforded by a comparison of the Dome and Hollinger practice. The advantages claimed for stamping in water and amalgamating are a better recovery of the coarse free gold and the saving in treatment cost, due to a smaller amount of solution to precipitate and a smaller amount of precipitate to handle. At first sight it would also appear that a saving in dissolved losses from the filters would be made, owing to the smaller amount of gold in solution going to the filters.

While at the Hollinger there is a smaller percentage of amalgamable gold in the mill heads than at the Dome, on the other hand, the coarse free gold per ton of ore in the mill heads is about the same due to the fact that the head assay is nearly triple that of the Dome. Hence, if amalgamation is necessary in order to assure the dissolution of coarse gold in cyanide solution, difficulty from this source should be experienced at the Hollinger, where crushing is in solution with no amalgamation.

This is not the case, however; all the coarse gold goes into solution in the classifier-tube mill closed circuit. This is proved by two facts in connection with the Hollinger operations. Table concentration after fine grinding is practised at the Hollinger, and any coarse free gold passing the tube mill-classifier closed circuit would be caught on the tables. No coarse gold is present

in the table concentrates, however, no color of free gold ever showing on the tables; also practically no amalgam was produced (under one per cent of the total values recovered) during six months, pan-amalgamation of concentrates from the tables.

Facts also indicate that crushing in solution without amalgamation has the best of the argument in regard to amount of solution precipitated and dissolved gold mechanically lost. With the head assay at the Hollinger about three times as high, the precipitation ratio is only about twice that at the Dome, and the mechanical loss of dissolved gold and cyanide only about one-half. Nor is any saving in dissolved losses or treatment cost proved in favor of crushing in water followed by amalgamating. On the other hand, there are added to the treatment cost, (1) cost of amalgamation, (2) cost of increased cyanide consumption due to 'waste' solution precipitated and thrown away, and (3) in winter the cost of heating to mill temperature the quantity of water—equaling several times the weight of ore treated and introduced at nearly a freezing temperature—to replace the water and waste solution discharged with the tailing, which would not be necessary if crushing were done in solution. The loss of gold left in 'waste' solution after precipitation must also be added to the cost.

In comparing Dome and Hollinger metallurgical practice, it is only fair to state that since the Dome ore is harder and more compact, the gold may be less spongy and therefore less amenable to cyanidation. The only deduction which can be drawn from the facts, as far as I know them, is, that apparently in treating average-and low-grade ore of the district amalgamation can be eliminated; that if amalgamation is eliminated and solution introduced at the stamps, a considerable saving in operating cost, cyanide, and dissolved gold losses is possible

Unquestionably more extensive study has been given to the treatment of the Dome ore than to any other in the district. Hence one hesitates to make what may appear to be a criticism of an operating system probably justified by a careful balancing of co-ordinate factors by an eminent firm of metallurgists. However, no metallurgical discussion would be complete without touching upon this point, which is the salient difference between the two metallurgical systems of the district.

Concentration versus Non-concentration.—The Hollinger is the only mill making a table concentration. About 16 per cent of the gold in the ore is recovered in the concentrate, and the advantages claimed would indicate that the possibilities justify careful consideration. The pulp, with the concentrate removed, needs much less careful treatment than the entire pulp, concentrate included, would require, hence a small tonnage of concentrate may be given whatever treatment it demands to get the best result, while the large tonnage of pulp free from concentrate may receive the much smaller amount of attention it requires.

Table concentration at the Hollinger costs about five cents per ton of ore treated and recovers about eighty lbs. of concentrate per ton of ore, assaying about two and a half cents per lb., worth, therefore, about two dollars. In the careful treatment given the concentrate the value per pound of concentrate is brought down to about 0.3 cents; in other words, a saving of one dollar and seventy-six cents is made from the eighty lbs. of concentrate from each ton of ore, which is a large enough amount to warrant the expenditure of five cents to safeguard. The performance of the thickeners and filters is improved if a feed can be maintained composed of particles of one specific gravity, so that from the standpoint of better mill performance, due to keeping the concentrates out of the thickeners and filter, I am of the opinion that the expenditure of five cents per ton of ore is justified where the concentrates taken out amount to say four per cent or more of the total tonnage. Another advantage is that even after a very careful treatment the concentrate tailing assays from five dollars to seven dollars per ton of concentrate, equal to twelve cents to twenty cents per ton of ore. Also the saving in treatment cost, due to the less agitation required for the pulp freed from concentrate, should be credited to concentration. Hence, I should say that if table concentration cost twenty cents per ton, instead of five cents, it would still be justified when a considerable amount of Class B ore is to be treated.

Agitation.—The ore particles composing the pulp coming to the agitators are extremely quick-settling and largely granular; after only a few minutes' shut down the pulp compacts

solidly in the bottom of the agitator so that, mechanically, agitation is a difficult problem. At the Dome four Pachucas in series are in operation, but all the other mills use the Dorr agitator, which seems to be peculiarly adapted to the local requirements. From the trouble experienced in keeping the Hollinger pulp in suspension in the filter-loading vats I judge that the Pachuca agitator is expensive in power required to prevent the filling up of the cone. With a trifling amount of power and a normal air consumption the Dorr agitator meets all the mechanical difficulties.

Metallurgically the quicker-settling particles need longer treatment than the lighter material. Selective agitation of the quick-settling particles is therefore essential if the best results are to be obtained.[1] The Dorr agitator, allowing as it does control over the rate of flow through the tank of material of greater or less than the average settling rate, meets the metallurgical requirements of the ore very nicely.

Filter Methods of the District Compared.—The Merrill filter is in use at the Dome, and while direct treatment in the presses has been tried, it has been found that the use of agitators for the dissolution of the precious metals is preferable, as a very large and expensive filter installation would otherwise be required. The Moore filter at the Hollinger has indicated that the ore contains such a large proportion of quick-settling material that vacuum filtration is not altogether satisfactory. In the loading vats, six air lifts are used, requiring forty h.p. at the compressor, and even with this intense circulation the heavy slime accumulates on the 60° hopper bottoms, resulting in such damage to the leaves that about one-third of the filter-operating cost is represented in repairs to filter leaves. Then, too, with the strong circulation, due to the air lifts, the cakes are channeled and uneven. On the whole, I do not consider that vacuum filtration is adapted to the Porcupine ore.

At the McIntyre mill a Burt filter is doing very good work and is stated to have been entirely satisfactory. At the Hollinger and the McIntyre mills, the pulp from the new sections will

[1]For an exposition of the term 'selective agitation,' see the paper by J. V. N. Dorr, Bull. A. I. M. E. No. 92, page 2072 (Aug. 1914).

be treated by continuous counter-current decantation, while the pulp from the original sections will continue to be put through the filters, so that shortly some interesting comparative figures on the two methods should be available.

Continuous Counter-current Decantation.—It has been previously mentioned that no oxodized ore occurs in the district, and the clean undecomposed rock breaks down to give an ideal product in the thickeners. Class A ore makes no colloid, and Class B ore, while grinding to an extremely fine, amorphous product, gives little trouble in settling, owing to its high specific gravity. Class A ore can be thickened to thirty per cent moisture and Class B to thirty-five to forty-five per cent, depending upon the percentage of concentrate. The critical moisture is forty-five per cent., when five per cent of concentrate is present, and about thirty-five per cent. moisture with Class B pulp, free from concentrate. On account of being able to get such unusually low moistures in the thickeners, a very high recovery of dissolved metals is possible by continuous decantation. Also, the fact that the cyanide strength of the solution from agitators to thickeners need only be carried at slightly above one pound per ton favours the decantation system, where the mechanical loss of cyanide is generally higher than in ordinary filter practice. At the Porcupine Crown, with about thirteen dollars going into solution per ton of ore and using four steps of decantation, with no filter, the dissolved gold loss is only five cents, and the mechanical loss of cyanide only 0.32 lbs. per ton of ore.

The Hollinger mill put in two steps of continuous decantation early in 1913, and the complete counter-current decantation system was installed in the cyanide extension of the Porcupine Crown later the same year. Later still, the Vipond installed the counter-current decantation system, as did the McIntyre when the mill was enlarged. The Hollinger has a complete three hundred ton plant under construction and the Acme mill a six hundred ton plant, both to use this system.

Mill Design.—The proper design for a mill treating Porcupine ore will depend upon the proportions of Class A and Class B ore to be handled. Unless there is to be a large excess of Class

A ore, amalgamation may be dispensed with, as the recovery by amalgamation will not warrant its use. If Class A ore is in large excess it would still be an open question, but from a recovery standpoint amalgamation is unnecessary.

The Hardinge ball mill may not show up as well on Class A as on Class B ore, but I am inclined to think that it would. With an excess of Class B ore the ball mill will be superior to stamps. I am of the opinion that a cylindrical tube mill should be used for fine grinding, rather than a conical mill, if only for a theoretically better dissolution of coarse gold.

For the treatment of any considerable proportion of Class B ore, table concentration, with separate treatment of the concentrates, will probably pay.

Agitation should be arranged to be continuous, preferably in a series of flat-bottomed agitators, allowing a preferential treatment for the quicker-settling portion of the ore.

If filtration is used, a pressure filter will be more satisfactory than a vacuum filter; however, the ore is so perfectly adapted to continuous counter-current decantation that this would seem to be the proper treatment.

On account of the severe winter conditions and the high cost of fuel, the object to strive for in the design should be as compact an arrangement of the equipment as possible, so as to minimize the cubic area of buildings to be heated.

In the district, the water supply is ample, the sites for mills are good, and the facilities for convenient tailing disposal are adequate.

COTTRELL PROCESSES OF ELECTRICAL PRECIPITATION

By Walter A. Schmidt, Los Angeles, Cal.

Annual Meeting, Toronto, 1915

The subject which I have the honour to present is the condensation of fume, smoke and dust by electrical means, or what is generally referred to as the Cottrell Process of Electrical Precipitation. This subject, however, has recently come into prominence as the direct result of numerous practical installations of sufficient size to eliminate the argument of experiment versus practice. My excuse for presenting this subject anew to this Institute at this time is the interest which most men of practical experience have in the problem of converting scientific truths into commercial processes. A hurried review of the development work conducted during the past eight years and which led to the practical establishment of these processes may, therefore, bring out some matters of interest to the members of the Institute.

The general problem of the removal of suspended matter from gases has been growing in importance for a number of years. On the one hand, this has been occasioned by the increasing seriousness of the pollution of the atmosphere with its subsequent hygienic and economic damage; on the other hand, through the ever increasing demand for greater economy in our industrial processes. This problem has grown steadily with the growth of the factory and the increase in volume of the furnace gases emitted from the exit stacks. In some of our larger industrial centres such furnace gases are of sufficient volume that the ordinary movements of the atmosphere are wholly incapable of effecting satisfactory dilution when these gases contain either poisonous constituents or when they carry large amounts of suspended matter. Then, as the result of keener competition, coupled with the substitution of lean raw materials for the richer ones formerly available, it is becoming more and more important to eliminate losses in all industrial processes. Those occasioned through the loss of materials swept along by the furnace gases,

and out through the exit stack, in many instances compose a large portion of such total losses. Furthermore, the problem has specific interest through the possibilities offered for improvement in industrial processes when gases can be cheaply and effectively cleaned of the suspended particles carried by them.

The removal of suspended particles from gases is a purely mechanical problem, even though the individual particles are exceedingly small, and practically all mechanical forces have been pressed into service in dealing with this question. The various forces have led to the development of as many different processes, each having its particular field of application. These processes may be roughly grouped as follows: Settling Processes, which will include settling flues, expansion chambers, wire chambers, baffle houses, etc.; Filtration Processes, covering bag-houses, filtration towers, etc.; Scrubbing Processes, including all forms of washing towers and mechanical scrubbers, such as the Thiesen, Schwartz-Bayer or Feld types and also including injector scrubbers; Centrifuging Processes, embracing ordinary cyclones, complex cyclones such as the Dyblie Whirler, mechanically revolving whirlers; and Electrical Precipitation, with which in this paper I propose to deal exclusively.

The electrical precipitation processes give us perhaps the only means for taking hold of the individual particles of suspended matter. During the process of treatment the individual particles are given separate electrical charges and are then caused to migrate under the electrical forces out of the current of advancing gases. The force which can be exerted upon the individual particle depends upon the construction of the apparatus as well as the electrical characteristics of the gases under treatment. It can readily be seen, however, that a process embodying such selective action between the suspended matter and the gases has a tremendous advantage, when large gas volumes are to be treated, over all processes based upon a mere difference of some specific physical characteristic, as, for example, the difference in inertia as made use of in baffle houses, or the difference in specific gravity as utilized in settling chambers and centrifuging processes. In all such processes the gases as well as the suspended particles must be treated, while with the

electrical process it is merely necessary to exert a sufficient force upon the individual particle to overcome the buoyancy and carrying power of the moving gases.

The operation of the electrical processes, when analyzed into its simplest factors, may be described as follows: The gases, along with the suspended material, are passed into an apparatus comprising a system of electrodes. These electrodes are of two types, one type of a form facilitating electric discharge from the surface thereof, the other type of such form as to minimize or prevent discharge from its surface. These electrodes are so placed in the apparatus that the two types oppose each other and a silent or glow discharge is maintained between them, by impressing a sufficiently high unidirectional electric potential upon them. The gases are passed through this electric discharge, which may be viewed as a constant stream of ions, or particles of electricity, flowing from the discharge electrode to the non-discharging electrode. As the suspended particles come into this stream of ions they are bombarded by them. Those ions which strike the particles are stopped in their passage from electrode to electrode, attach themselves to the particles and give up their electric charge to them. In this manner the particles gather a sufficient electric charge to move under the force of the electric field between the electrodes, then migrate toward the electrode and are deposited thereon while the gases pass on unaffected. In reality, the principle cannot be so simply carried out and a great deal depends upon the proper design of the apparatus. The composition of the gases, the nature of the suspended particles, the temperature and many other factors must be taken into consideration in designing a plant to handle any specific problem.

Many undoubtedly will be familiar with the old laboratory experiment of blowing out a candle by means of the electric windage flowing off of a needle point highly charged with electricity. This windage is caused by the repulsion of the gas molecules from the charged point in a manner similar to the method just described. Gases themselves can be made to move under the electric discharge as the result of the charging of the gas molecules and the bombardment of the molecules by the constant stream of ions, composing the discharge. This action,

however, is not nearly so pronounced as the translation of the larger suspended particles, which can be translated with respect to the gases with remarkable rapidity.

Again referring to the electrified needle point. If this be given a unidirectional charge, for example by a Wimshurst machine, and an insulated body be brought close to the point, it instantly takes up an electric charge similar in sign to the charge upon the needle point. If such insulated body is free to move, as for example a pith ball suspended on a silk thread, it is repelled from the point. If now, such electrified-point is placed opposite to a grounded plate and a freely floating particle passed into the discharge between the point and the plate, such particle acts similarly to the pith ball, takes up an electric charge similar in sign to the charge upon the needle point, is repelled therefrom, and upon striking the grounded plate, gives up its electric charge to the plate and is deposited thereon. This is precisely what takes place in an electrical precipitation apparatus, or so-called treater, the difference being mainly constructional details, the choosing of proper electrodes and the substitution of reliable and and adequate electrical equipment.

The removal of suspended particles from gases by electric means is quite an old idea, it having been quite independently suggested by a number of investigators at intervals of about twenty-five years for the better part of a century. It seems that the method was first suggested as a means for suppressing smoke by Hohlfeld in 1824. In an article entitled "The Precipitation of Smoke by means of Electricity" he refers "to the increase of rain and hail after a flash of lightning, and describes how he filled a globe with smoke and led into it a pointed wire connected to an electric machine, which caused the smoke to settle." It was again suggested by Guitard in 1850 in an article in the *Mechanics Magazine*. Reference is even made to this article in Wiedemann's Galvanismus. This early work and the suggestions of Hohlfeld and Guitard seem to have been quite forgotten, as nothing ever came of them as far as any records show. During the period roughly designated as twenty-five years following the appearance of Guitard's article, a large amount of work was conducted by Crookes, Thompson, Aitken, Lodge, Tyndall and others upon molecular bombardment.

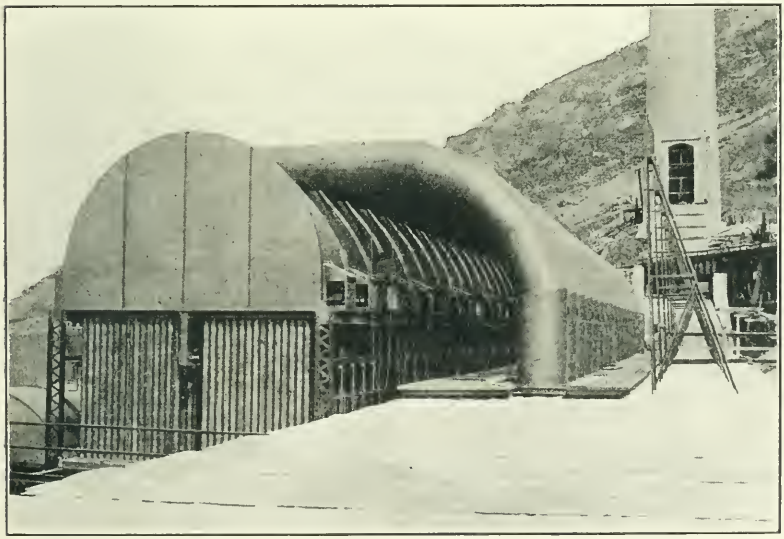
They investigated the repulsion of dust from heated bodies, the force exerted by the bombardment within a Crooke's tube and similar phenomena. Oliver Lodge then again brought the subject of electrical precipitation into light in a series of articles appearing in 1884, 1885 and 1886, the main article being published in the Journal Soc. Chem. Ind., 1886, this giving special discussion of the possibilities of electrical deposition of suspended matter from gases. The early work of Hohlfeld and Guitard seem to have remained entirely forgotten until brought to light again in 1905 by Sir Oliver Lodge in an historical note published in *Nature* in that year.

The first attempt to apply the processes commercially seems to have been made at the Dee Bank Lead Works and the first patents on the process were granted to Mr. Alfred O. Walker of that firm. He obtained patents in various countries during the years 1884-1886. All of these patents, however, have long since expired. The apparatus was installed in 1885 by the works manager, a Mr. Hutchings, with the co-operation of Prof. Lodge. It was described at the time as "consisting of a system of metallic points situated in the flue from the lead furnaces and excited from two Wimshurst machines, each machine driven by a one-horse power steam engine." The apparatus was unsuccessful and nothing further seems to have been done with the process by these early pioneers. Quite independently, Dr. Karl Möller of Germany also suggested this method for smoke abatement and he secured a German patent in 1884. He, however, did not develop any commercial application, but his son, Erwin Möller, has done considerable development work on the processes and is now actively engaged with us in the prosecution of the work upon the continent of Europe. In 1900 Eissler again called attention to the work of Walker and since then an occasional publication has kept the process in the public eye.

Our work commenced in 1906 when Dr. F. G. Cottrell investigated various problems in connection with the Contact Process of sulphuric acid manufacture. Dr. Cottrell was at that time a Professor at the University of California. He is now Chief Chemist of the U. S. Bureau of Mines and is no longer interested commercially in the processes which bear his name. In 1906 Cottrell was confronted with the problem of removing

sulphuric acid mist from gases before these went to the contact towers. He repeated the early experiments of Lodge and became convinced of the possibilities in the method. The first large scale tests were made at the Hercules Works of the E. I. du Pont de Nemours Powder Company at Pinole on San Francisco Bay, where the gases from a Mannheim Contact Sulphuric Acid unit were employed. These tests, although by no means complete, demonstrated conclusively the commercial possibilities of the process. Since then the work has largely been the development and reduction to sound engineering practice of the fundamental principles disclosed in this early work. The failure of the early commercial work of Lodge and Walker can without doubt be largely attributed to the unsatisfactory and unreliable electrical equipment then available. Modern electrical apparatus has made the development of these processes a much easier task than that which confronted the early pioneer investigators.

A full bibliography of the early work is contained in an article by Cottrell entitled "Problems in Smoke, Fume and Dust Abatement" published in the Report of the Smithsonian



Cottrell Electrical Precipitator for the collection of copper convertor fumes.

Institution of Washington for the year 1913, which, however, has just recently appeared in print.

The first commercial application of our processes was made at the works of the Selby Smelting and Lead Company on San Francisco Bay. This company was threatened with injunction suits on grounds of the alleged damage done by the sulphuric acid escaping from kettles of boiling sulphuric acid in which the silver was parted from the gold in the alloy coming from the cupels. The striking results obtained in the experiments at Pinole in the precipitation of sulphuric acid mist, drew the attention of the Selby Company and, after a few successful preliminary tests, a treater was constructed in 1907. This treater has been in constant service ever since and several duplicate treaters have been constructed elsewhere. This apparatus treats approximately four thousand cubic feet of gas per minute. The apparatus is constructed in a flue having a four by four foot cross-section. The collecting electrodes are lead plates four feet long and four inches wide, spaced five inches apart. The discharge electrodes consist of serrated micanite clamped between strips of hard lead. The electric power is taken from the 460 volt, 80 cycle factory power line, transformed up to 17,000 volts and rectified by means of a synchronous contact maker, the resultant intermittent direct current energizing the discharge electrodes. The power consumption of the complete apparatus, including the driving of the rectifier, is two kilowatts. No additional men are employed to operate the apparatus; and full operating costs, including power, repairs and attendance is approximately twenty dollars per month. The acid collected is of 40°B. strength and when the kettles are boiling vigourously, over two gallons per minute of such acid is collected. As far as any records show, this was the first commercially successful electrical precipitation apparatus ever constructed.

Following upon this work a series of tests were made upon the roaster gases at the same works. An experimental treater was constructed which handled the full fifty thousand cubic feet of gas per minute, passing in the flue. The general design of this treater was similar to that of the parting flue apparatus just described. Fundamental changes in metallurgical practice

at the smelter made the installation of the permanent equipment unnecessary.

The next installation was at the Balaklala Smelter at Coram in Shasta County, California. This smelter was in the heart of the Shasta copper belt, together with the Keswick smelter, the Bully Hill smelter and the smelter of the Mammoth Copper Mining Company. As the result of litigation, certain restrictions were placed upon all of the smelters in this district, relative to fume control. These restrictions included the removal of all suspended matter from the exit gases and the dilution of such gases to prevent the sulphur dioxide content from exceeding three-quarters of one per cent. The Bully Hill and Keswick smelters suspended operations, the Mammoth smelter installed a bag-house and the Balaklala undertook the installation of the Cottrell Processes. This was the first large installation of these processes and it was necessary in the design of this apparatus to figure far beyond the experimental and test data then available. The gas volume was approximately 250,000 cubic feet per minute. The general design of treater was similar to that installed at Selby, except that iron was used instead of lead, the gases being non-corrosive. Nine units of ten by ten feet cross-section were built in parallel, each connected to a common inlet and common outlet flue. The entire structure other than the flues was built of iron. The collecting electrodes were of No. 10 sheet iron, ten feet long and six inches wide, spaced five inches apart. The discharge electrodes were made of asbestos fibre intertwined between iron wires. Serrated micanite twisted between iron wires was also used. This apparatus was operated on about twenty-five thousand volts. The collected fume was dry and adhered to the electrodes. To remove this collected material from the electrodes, the unit to be cleaned was by-passed and the electrodes given a mechanical jarring or rapping, the dust falling into hoppers from which it was removed by means of screw conveyors. Each unit was cleaned once in eight hours. During the cleaning of one unit the remaining eight treated the entire volume of the gases. The total cost of the installation, including flues, fans, motors, and electrical equipment was one hundred and ten thousand dollars. The power consumption was about 120 kilowatts. The attendance was one man per

shift to attend the electrical generating apparatus and two men and a foreman per shift to look after the cleaning of treaters and the removal of the precipitated fume. The amount of material collected was eight tons per twenty-four hours, this being a very fine dust high in copper and zinc, but was not of particular commercial value due to its high zinc content. The entire smelter was closed down in July, 1911, as the result of litigation troubles, it being decided to discontinue operations until such time as a practical method could be developed for removing sulphur dioxide from the gases as well as the suspended particles.

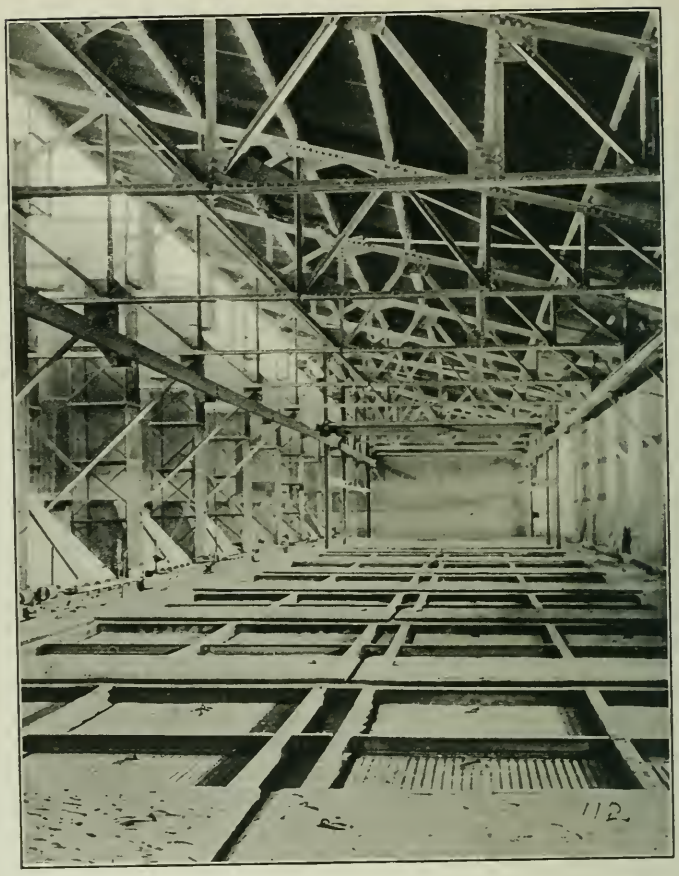
Riverside Portland Cement Co"s rotary kiln and stacks before installation of Cottrell dust precipitators.

Shortly before the work at Coram was discontinued, experiments were commenced at the plant of the Riverside Portland Cement Company on the problem of removing from the furnace gases the dust carried out of the rotary kilns used for the burning of cement. This problem was quite different from those upon which the processes had been tried up to that time. This difference was occasioned by the dryness of the gases, as well as

The Riverside Cement Works after installation of dust precipitators.

their high temperature. The enormous volumes of gases and the large quantities of dust which had to be handled also presented new problems. The Riverside Portland Cement Company, with its factory in the heart of the fertile orange belt of Southern California, had become the centre of serious agitation which threatened the factory with an ultimate injunction. Many demands had been met by damage payments and much land was purchased from complaining neighbours, but although the company was exceptionally liberal in meeting the demands of the growers, it became apparent that the dust had to be prevented from escaping into the atmosphere or the risk be run of having the operation of the factory discontinued by injunction.

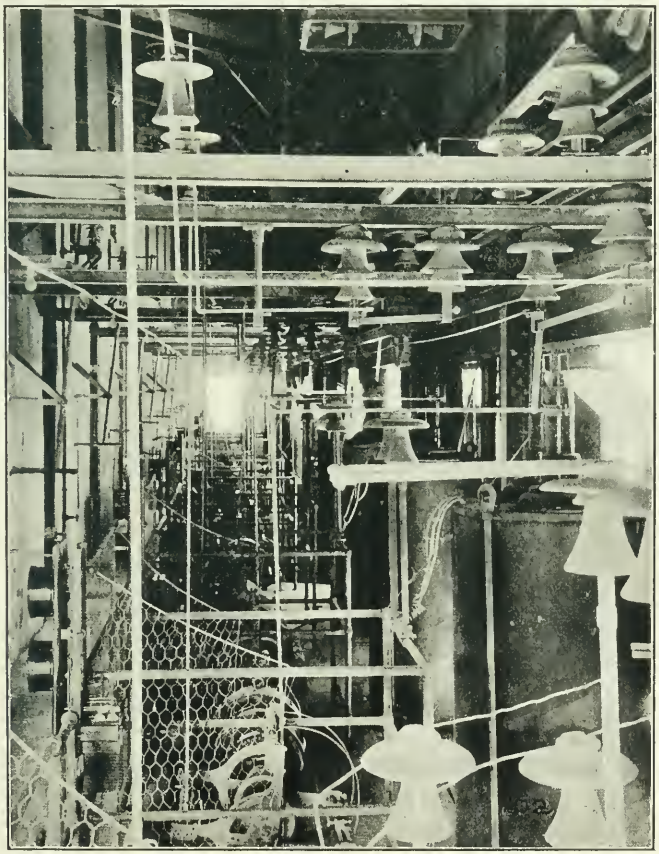
The experimental work at this factory included the building of several fair sized treaters, roughly seventy-five thousand dollars having been expended upon the preliminary tests. These, however, proved successful. The final installation was put into



A view across the top of the Cottrell precipitators at the Riverside Cement Works.

operation January 1913 and has been in continuous operation ever since. This installation treats almost one million cubic feet of gas per minute, the temperature being four hundred and fifty degrees Centigrade. The amount of dust collected averages about ninety tons per twenty-four hours. The total power consumption of this installation, including the motors driving the rectifiers and the screw conveyors removing the collected dust is forty kilowatts. The efficiency of collection of the suspended particles averages ninety-six per cent the escaping material being largely potassium and sodium sulphate. The heavy dust is all

collected in the treater. The attendance consists of one operator and a helper per twelve hour shift. The cost of this installation was high, for the reason that there was no space available for the construction of the apparatus in the factory and the entire construction had to be carried over the building and placed upon a platform eighty feet above the ground. Each of the ten furnaces was maintained independent of the others and each was equipped with two treater units, each treater being sufficiently large to handle the entire gas volume coming from the corresponding furnace. This was done to insure absolute certainty of



Rectifier Room, showing transformers, mechanical rectifiers, and switchboard panels, at Riverside Cement Works.

operation, should difficulty arise with any one unit. The furnaces were kept independent of one another to give greater freedom of operation, independent stacks being the universal rule in cement factories in this country. The entire structure, including electrical equipment cost $165,000 of which over half went into the expensive platform substructure and housing.

This photograph represents the quantity of dust collected in five days by Cottrell precipitators attached to ten rotary kiln stacks at the Riverside Cement Works.

The details of design of this installation vary considerably from the Selby and Balaklala treaters. The collecting electrodes are made of wire mesh reinforced with iron angles and are spaced twelve inches apart. The discharge electrodes are iron wire. The operating potential is 45,000 volts. The removal of dust from the electrodes is effected by a simple mechanical system, the electrodes being jarred by means of air hammers and the dampers operated by means of air cylinders, all controlled from the operating platform. The dust falls into hoppers and runs to screw conveyors which feed into automatic scales

and from there it is dumped into a general cross conveyor which carries the dust into the bins. The treaters are cleaned once every hour, the cleaning of the entire twenty units requiring less than five minutes.

Recently an additional kiln was installed at this factory, in which all of the collected dust is now burned to cement. Formerly the dust was disposed of as a soil dressing. The dust is partially calcined raw mix, containing no cement, but carries about two per cent of potash. This material makes a splendid dressing for certain soils. With the present practice of making cement from the dust, the potash is concentrated and the dust caught from the new kiln has a potash content varying from twelve to twenty-one per cent K_2O, depending on whether the kiln burns all dust or part dust and part raw mix. The general treating system does not supply sufficient material to permit operating this kiln exclusively upon dust.

There is one interesting feature in connection with this installation. Although the main equipment was installed to guard off injunction proceedings and the new kiln was installed largely to dispose of the collected material, as the location of the plant made it inconvenient to dispose of the dust for soil dressing purposes, still the final result was the first potash recovery plant ever put into commercial operation. The raw mix at this plant is composed of limestome and a decomposed granite high in feldspathic material. This decomposed granite contains seventy-five hundredths of one per cent K_2O. The granite forms one quarter of the raw mix, giving a K_2O content in the raw mix of less than one-fifth of one per cent. The potash content in the dust from the main treating system averages two per cent, while the final product exceeds twenty per cent K_2O when dust is burned exclusively in the concentrating kiln. Four to five tons of this concentrated material are caught per twenty-four hours. This material is used at present as a soildressing in the groves and on the farms owned by the Cement Company. Although these results may not be striking from the viewpoint of independent potash manufacture, they serve to demonstrate beyond question the possibilities in the recovery of potash from feldspar, particularly if this can be carried on as a by-product process to cement

manufacture. Vast deposits of feldspar are available, averaging as high as ten per cent K_2O, while the Riverside material only runs three-quarters of one per cent.

Until the completion of the Riverside installation little attempt was made to advance the commercial progress of the processes, it being considered sounder policy to work out construction details in a few installations close to headquarters. During the early part of 1912 Dr. Cottrell became the moving spirit in the formation of the Research Corporation which is, so to speak, a subsidiary of the Smithsonian Institution and he liberally donated to it all rights to his processes in the United States east of the Rocky Mountains excepting the application to the cement industry. The latter rights and the rights in the Western Territory of the United States had previously been acquired by the interests now composing the Western Precipitation Company. This latter company also holds the rights to the processes in Canada and Mexico.

After the completion of the Riverside installation and the formation of the Research Corporation, general prosecution of the commercial development of the processes was undertaken. Since then progress has been rapid and the scope of the work has been broadened immensely. The following are some of the more important applications of the processes.

At the Garfield smelter near Salt Lake City the gases from the copper convertors are treated in a precipitator of a different type from those just described. The collecting electrodes are cylinders five inches in diameter, the discharge electrode being suspended along the axis of the cylinder. We differentiate between this type and the Selby type of treater by terming one "multiple pipe treater" and the other "plate treater." The Garfield tests were commenced in 1911 and the final installation was put into operation in 1914. The final installation consists of seven sections each having its separate electrical energizing set. Each section is composed of 360 five-inch iron pipes ten feet long, a total of 2520 pipes in all. The total volume of convertor gas is being treated in this installation, being approximately 200,000 cubic feet per minute. In contrast with the Selby, Balaklala and Riverside treaters, the Garfield installation was built entirely to effect a saving of the copper

and lead losses from the convertors. The efficiency of collection of the suspended matter averages about ninety per cent. The installation, including the electrical equipment, but exclusive of the flue connections cost approximately fifty thousand dollars. The power consumption is roughly fifty kilowatts and one operator and a helper are employed per shift.

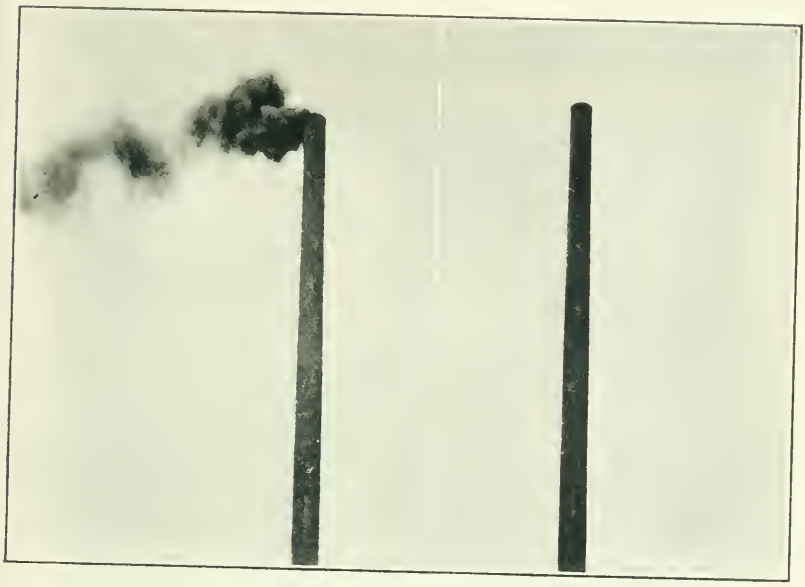
The largest installation for metallurgical use is now under construction near Globe, Arizona, at the new Inspiration smelter of the International Smelting and Refining Company. Two complete treater installations are under construction at this plant, The largest of the two will treat the gases from the copper convertors, the gas volume being estimated at about 300,000 cubic feet per minute. The second installation will handle the gases from the dryers. The latter is a new problem of special interest. At the Inspiration and the Miami mills the table middlings will be treated by the Flotation Process to recover the finely divided sulphides. The flotation concentrates will be filtered in Oliver filters and dried in Wedge furnaces, then sent to the reverberatories. Each Wedge furnace will be equipped with a separate electrical treater, placed almost directly over the furnace, and each treater will discharge through a separate short stack into the atmosphere. The treater, therefore, will virtually be a part of the stack. The treaters are being installed for the purpose of saving the fine particles of dust which will be carried out of the dryers by the draft. It is estimated that a considerable quantity of dust will be carried from these dryers, due to the extreme fineness of the flotation concentrates. As neither the convertor nor dryer installations are completed, no information on costs or operating conditions is available. The plant will be put into operation within the next sixty days unless unforeseen delays occur. Both convertor and dryer installations are of the multiple pipe type, the pipes being twelve inches in diameter.

Another multiple pipe treater is in operation at Trail, B.C., at the smelter of the Consolidated Mining and Smelting Company of Canada, Limited. This treater has been installed for the purpose of saving the lead losses from the lead blast furnaces. It consists of 384 twelve-and-one-half inch, inside diameter, pipes, fifteen feet long. The installation is divided

into twelve sections of thirty-two pipes each, all operating at present on one electrical energizing set. This treater is somewhat over-loaded at present and an additional six sections will be installed in the immediate future. Approximately one hundred thousand cubic feet of gas per minute is being treated in this installation, at a temperature varying from two hundred to three hundred degrees Fahrenheit. The gases come from the three new lead blast furnaces recently installed at this smelter. The dust collected at present in the treater, exclusive of the connecting flues, averages about six tons per day. The collected dust averages sixty-five per cent lead and carries a small amount of silver. The lead is present as oxide, sulphide and sulphate, in varying proportions depending upon furnace conditions. This fine dust readily sinters, it being sufficient to place the dust in brick lined cylinders having a perforated bottom and open at the top. Sintering is started by local heating with a few sticks of wood and air is then blown through the mass. The sintered material is sent to the blast furnaces. The Trail installation consumes about ten kilowatts of electrical energy and one operator per shift is sufficient for complete attendance. This treater differs from other newer treaters in that no mechanical cleaning system has been installed, the management considering that the operator would have insufficient work if the task of rapping the pipes were eliminated. The cost of this installation cannot be readily ascertained as it composed a part of extensive reconstruction work affecting the entire blast furnace department. It is estimated that the installation complete, exclusive of flue connections cost between thirty and thirty-five thousand dollars. The operating potential of this treater is seventy-five thousand volts.

A second treater is now under construction at Trail, this to treat the combined gases from two Wedge furnaces, one Dwight Lloyd sintering machine and thirty-six Huntington Heberlein pots. It is also planned to install an electrical treater at the refinery at Trail to collect the fumes from the Dore furnace and the melting pot. This fume carries considerable silver, some gold and about fifty-five per cent antimony. The metallurgical problem will be to separate the arsenic from the anti-

mony after the fume is collected. Other projects are also in contemplation at Trail in the field of electrical precipitation.

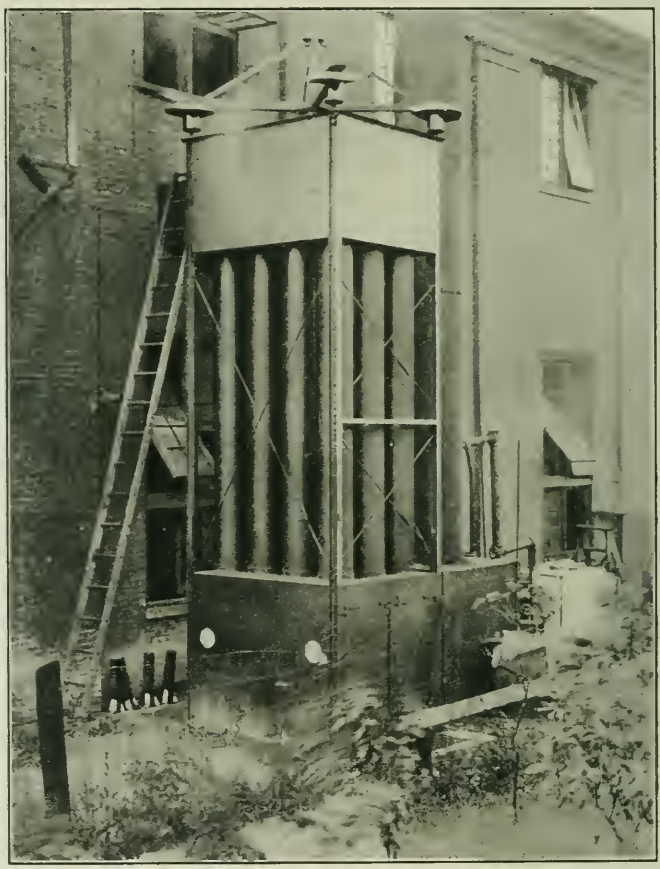


Appearance of a boiler stack with the current off and on.

A problem similar to that of the Trail refinery has been met by a treater installation at the refinery of the Raritan Copper Works at Perth Amboy, New Jersey. This installation is of particular interest in that the treater was installed beyond the water scrubbers, which were formerly assumed to give satisfactory cleaning of the gases. The values collected, however, are sufficient to form a large item of saving, particularly as the operating costs are very low. The electrical energy consumed is less than three kilowatts and no special attendants are required.

Numerous other installations have been made at metallurgical and chemical works for permanent use and some for experimental purposes. It is sufficient to state that each particular installation requires a rearrangement of parts to meet structural requirements, or a readjustment of electrode systems, change in construction details and modification of

electrical equipment to meet specific conditions determined by gas composition, fume composition and temperature. To show the broad variety of problems which these processes have been called upon to solve, a brief reference to some of these installations may be of interest.



Pipe type of Cottrell Precipitator at Pittsburg, Pa. This precipitator cleans the gases from an 80 h.p. coal-fixed boiler.

At the Omaha refinery of the American Smelting and Refining Company, a treater has been installed for the collection of sulphuric acid mist, this being similar to the first treater

installed at the Selby Works. A similar installation is to be made at the Baltimore Refinery.

At Tooele, Utah, a treater has been installed to treat the gases from two Dwight Lloyd sintering machines, a total of twenty thousand cubic feet of gas per minute. The collected fume is mostly lead oxide but carries considerable elemental sulphur. This installation is to be enlarged to embrace the remaining eight sintering machines.

At the Goldschmidt Detinning Works, near Chicago, the processes are collecting tin oxide; at the Vulcan Detinning Works near New York experiments were made showing the feasibility of collecting hydrated tin chloride. At the Kings County Foundry a mixture of tin oxide with lead and zinc oxide was collected, this fume resulting from the furnacing of old tin scrap in a cupola furnace in smelting these for the manufacture of sash weights.

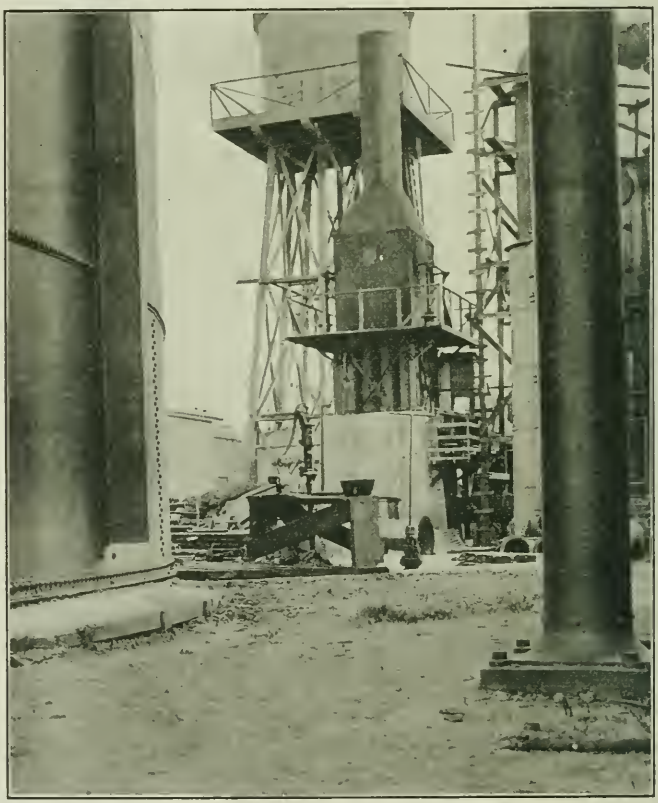
Dust is being removed from hot Wedge furnace gases prior to their entrance into the Glover tower at the sulphuric acid works of the Chemische Fabrik Griesheim near Frankfurt, Germany. In this installation the removal of dust is practically complete and the quality of acid produced has been considerably improved and the troubles in manufacture caused by the choking of the Glover towers has been entirely eliminated. This installation treats three thousand cubic feet of gas per minute at a temperature of 450 degrees Centigrade. It consumes less than one kilowatt of electrical energy and requires no special attendance. About four hundred pounds of dust are caught per twenty-four hours, this being mostly iron oxide, containing a small amount of arsenic. Similar, but larger installations of this kind are now under construction at the Chemische Fabrik Bochum near Hanover, and at the Elberfelde Farbenfabrik near Cologne. Tests on a similar problem are also under way at the works of the Graselli Chemical Company, at Cleveland, Ohio.

Sulphuric acid mist will be removed from roaster gases before these are discharged into the atmosphere, in a treater now under construction at the smelter at Sulitjelma, Norway. Similar work is contemplated in Arizona where this problem is of special economic value. If the large quantities of acid

now escaping with the roaster gases can be collected cheaply, it can then be used for leaching the low grade oxidized copper ores so abundant in that section of the country.

A treater is now being installed at the copper works at Duisburg near Dusseldorf for cleaning the combustible gases coming from the iron blast furnaces, prior to burning these gases under the boilers. Tests on a similar problem are contemplated at the works of the Bethlehem Steel Company at Bethlehem, Pennsylvania.

An extensive series of tests are now being conducted at the gas works in Vancouver, B.C., in which one step is the de-



Cottrell Electrical Precipitator at an illuminating gas works, for the removal of tar and lamp black from oil-fired gas machines.

tarring of illuminating gas by means of the electrical processes. This has been entirely successful. Similar work has been conducted at the Ann Arbor Gas Works by Prof. White.

Carbon and lamp black have been collected from boiler gases at the experimental boiler plant of the Bureau of Mines at Pittsburgh and from the generator gases at the gas works at Santa Rosa, California.

Naphthalene and other heavy hydrocarbons are being collected at the gas works at Portland, Oregon, in the briquetting plant, where the lamp black from the petroleum gas generators is dried and then pressed into briquettes for fuel purposes.

Tests are in progress at the Nipissing mine at Cobalt, Ontario, for collecting mercury fumes arising during the refining of the amalgam sponge.

A treater has been installed at the plant of the Universal Portland Cement Company near Chicago: a problem similar to that of the Riverside plant. This treater was installed for test purposes, but handled all of the gases from one large kiln, approximately one hundred thousand cubic feet per minute, operating under excessive temperature strains.

Dried milk, dried eggs and similar substances, manufactured by the spray drying process, have been successfully collected and a treater has been installed for the Milk Flour Company at Chicago.

Sugar dust has been collected in a test treater at the factory of the Californian Hawaian Sugar Company at Crockett, California.

Hydrochloric acid fumes are being collected at the North Works of the American Steel and Wire Company at Worcester, Massachusetts.

An installation is being constructed for the collection of of zinc oxide at the works of the Chase Rolling Mill Company.

Other materials have been worked upon experimentally, but this list of commercial applications well illustrates the wide range of possibilities which these processes possess.

In closing, I merely wish to touch upon two new features of the processes which present particular interest, as they

open new fields of industrial development and modification of present methods in industrial practice.

The first is fractional precipitation with temperature control. The mere fact that the electrical precipitation processes can effect a complete separation of suspended matter from gases at various temperatures, at once opens the possibility of separating two or more volatile substances from each other, provided such substances condense at different temperatures, from the gases in which they are carried. For example, should it be desired to separate arsenic from a copper bearing flue dust, all that will be necessary is to pass the gases through a treater at a temperature above the condensing point of the arsenic and precipitate the copper bearing flue dust. Should it then be desired to collect the arsenic, the gases can be cooled by radiation or through admission of cold air to a temperature at which the arsenic will form a fume, then pass the gases through a second treater and precipitate the arsenic. This problem has been operated at Anaconda and a permanent installation is to be constructed immediately. Here the collected flue dust was heated in a Godfrey roaster to drive off the arsenic. In the process of heating, considerable dust was carried along with the furnace gases. These gases were then passed through an electrical treater while hot and all of the dust precipitated. The gases were then cooled by the admission of cool air and the resultant fume of arsenic precipitated in a second treater. By this method, arsenic was collected as pure as 99.8 per cent As_2O_3. The problem is of particular importance at Anaconda where it is proposed to collect all of the fume now escaping from the main stack. The large amount of arsenic in the fume makes it imperative that the arsenic be eliminated from the flue dust before this is returned to the furnaces in order to prevent an arsenic circulation between the furnaces and the precipitator and an accumulation of arsenic beyond a controlable limit. It is proposed to eliminate the arsenic by the method just described.

Another interesting application of the same principle is the separation of sulphuric acid from flue dust, for example in roaster gases, which always carry some flue dust and usually considerable SO_3 from the conversion of the SO_2 through catal-

ysis by the iron oxide in the furnace. To obtain this acid it should only be necessary to remove the dust from the hot roaster gases, then cool and humidify and collect the resulting mist of sulphuric acid. It is proposed to conduct commercial tests upon this problem in the near future in the southwestern territory.

The advantages of such series treatment will immediately be apparent in connection with such processes as the volatilization of metals for recovery of such metals from low grade or complex ores. Such series treatment will not only allow of recovery of the valuable constituents but opens possibilities for concentration of same as well. Experimental work in this field is in progress, but this has not advanced sufficiently far to permit of published statements.

The second new departure is the precipitation of gaseous constituents, by intimately mixing with the gases certain finely divided materials which will react with or absorb the gaseous constituent which it is desired to remove, then precipitating the suspended particles which will then carry with them the absorbed or combined gas. An interesting application of this process has been made at the Hooker Electrochemical Works at Niagara, New York, in the bleaching powder department. The absorption towers, in which the chlorine is absorbed for the manufacture of bleaching powder, did not effect a complete absorption and small amounts of chlorine were carried into the atmosphere and constituted a nuisance. Hydrated lime dust was then blown into the flue carrying the exit gases. This dust absorbed the chlorine, and the lime dust together with the absorbed chlorine is separated from the gases in an electrical treater. This apparatus has been in operation for over a year and has given entire satisfaction. About thirty thousand cubic feet of gas is treated per minute and the chlorine effectively removed so that the exit gases are now absolutely non-irritating.

Such combination processes will without doubt prove of value in various chemical industries in simplifying many processes where large volumes of gases must be brought into intimate contact with solid or liquid reagents.

THE HALL PROCESS FOR RECOVERING SULPHUR

HOWARD F. WIERUM

Annual Meeting, Toronto, 1915

The Hall Process for recovering sulphur that is usually wasted may perhaps claim to be the most modern effect that has been made in this direction. Its full development is still in the making, and its complete possibilites are yet unknown.

The function of the process, which has attracted the most attention from the technical world, is not primarily sulphur manufacture, but rather smelter fume abatement,—or, to be more accurate, roaster fume abatement. Heretofore only spasmodically and locally has this discharge of sulphur dioxide into the air been considered in any other light than that of a nuisance. With but one or two exceptions, the ways and means considered for overcoming this nuisance have been either destruction of the fumes, or, what seems still more extravagant, the deliberate purchase of neighbouring miles of territory in order thus to acquire the right to damage that territory unhindered.

The technique of the Hall Process, and the story of its development to date, have recently been described as fully as may be, and appear in the Oct. 3, 1914 issue of the *Mining & Scientific Press* of San Francisco, and also as a paper in the Bulletin of the Mining & Metallurgical Society of America, Number 76, Sept. 30, 1914. It is therefore not necessary to dwell at length on these features. The patents already issued cover the early conception of the process, and later patent applications, practically allowed but not yet issued, cover very fully and basically all the practical developments which the writer worked out on a couple of full size (18 ft.) McDougall roasters at Coram, California. It may be well, however, to summarize the technical features briefly. In doing so it should be explained that various reactions enter into our results.

The earliest conception of the chemistry of the process may be described as simple distillation of both atoms of sulphur

in FeS_2, the first, or 'feeble' atom being driven off by heat, and the second, or 'fixed' atom, being replaced in its connection with the iron by oxygen as a stable equivalent, precipitated, so to speak. The basis for this latter supposition was first found in the behaviour of various metallic sulphides, when heated in a small closed furnace by means of a distinctly reducing flame, the furnace being kept meanwhile, under an obvious back-pressure, thus excluding the possibility of any atmospheric air, or oxygen entering. Gas was used as fuel,—first low grade producer gas of 130 B.T.U. per cu. ft., and then ordinary city gas of 550–600 B.T.U.; and the air for incompletely burning this gas was furnished under pressure, through easily adjusted valves. Small quantities of live steam were admitted. Under this set of conditions, it will be seen that oxidation would seem out of the question, inasmuch as not sufficient oxygen was supplied to even burn the gas completely. Dense volumes of elementary sulphur rolled out of the little exhaust chimney, often mingled with raw illuminating gas, easily detected by the smell, and practically entirely free from SO_2. The operation was continued, with continuous SO_2 determination throughout, until the evolution of sulphur practically ceased, and the ore under treatment (a pyrites carrying 40% S) was sampled and found to contain from 3 to 5% S, and the iron to be a mixture of Fe_2O_3 and Fe_3O_4. Over and over again, this was tried, and a c.p. monosulphide of iron used in place of pyrites, with always the same result: no appreciable SO_2, dense volumes of sulphur, and a cinder varying in its S contents according to the thoroughness, and the time devoted to the operation. Where did the oxygen come from that the iron contained, and how did the sulphur escape oxidation? The fact should be recorded,—a fact which probably hides the reason why later tests on large tonnages of ore demanded somewhat different, or at least additional conditions, in order to get the same results, viz: no measurements of the *total* gas, and *total* air used in these first small furnace experiences, were made. All that was observed was that there was enough gas and air to give heat (about 800° C), and that there was *always* an excess of gas. The deduction as to the chemistry of the process then was, that nascent oxygen was supplied by

a decomposition of the steam, and that iron showed a greater affinity for nascent oxygen, at this temperature, and under these conditions, than sulphur exerted. A further supposition was entertained, that if the steam was thus decomposed, H_2S would undoubtedly be formed, and that H_2S would furnish an immediate and potent reducing agent for any SO_2 that *might* be formed. In fact H_2S was sometimes detected in the escape gases.

Let us leave these suppositions at this point, and go on as briefly as possible to the application of these first results to the commercial size tests made in California.

The operation of a little circular 18-inch brick-lined pot proved quite a different matter from the operation of an 18-foot, 6 hearth McDougall roasting furnace. In the first place, the heating of this large furnace, and the 25 to 40 tons of ore daily, in a wastefully reducing atmosphere throughout, involved an almost prohibitory expense. Even though the identical conditions and results could have brought about in the McDougall, as in the little pot, it would not have been commercially interesting if the fuel cost indicated a final sulphur cost greater than the sales price. The process to be valuable, must be more than scientifically interesting, it must be practically possible and profitable. A most disheartening lack of time obtained, during these California tests, making it impossible to go into the finer technique, and to advance step by step; and making it imperative to get some results that would indicate success both financially and metallurgically. Consequently, operations were headed straight for this goal, and a set of conditions hastily but successfully attained whereby the following points were demonstrated. *1st.*—A sulphur elimination from 10 mesh ore, down to 5½% left in calcines (starting at 40%); *2nd.*—Control of SO_2 in fumes down to 3-10% at *most*; *3rd.*—A tonnage capacity, for 10 mesh sulphides, approximating the tonnage capacity for straight roasting, under corresponding specifications as to cinder analysis; *4th.*—A fuel consumption well within economic limits, amounting to about 50c per ton of ore, when using low grade, rather heavy, and cheap oil. This fuel cost of 50c per ton of ore would amount to about $1.50 per ton of sulphur in the case of a 48% S pyrite roasted

to 4%, and to about $3.00 per ton of sulphur, in the case of a 30% S concentrate, roasted to say 10% S. Thus the cost, not only of fuel, but of all other items entering into the manufacture of sulphur in this way, is directly proportionate to the number of tons of ore required to make one ton of sulphur, and this of course differs widely in different problems. It happens that in the plant where these experiments were tried calcines were required with only about 5% of S remaining, because the ore was so low in copper, that the matte produced in the subsequent reverberatory smelting operation would have been too low grade for normal converting operations, if more S than this were introduced with the smelting charge. At other plants, 8% S, would be desirable, at still others, 12%, always commensurate with the copper tenor of the ore or concentrates. Nine per cent may be taken as an average requirement. In the California case the 5% requirement, and 4% desired, may fairly be termed abnormal, when reverberatory smelting is to follow; still, for several days, the calcines (10 mesh) were maintained at 3.5 to 4%, with 2.8% as the lowest figure recorded. This feature is dwelt on for the purpose of emphasizing the fact that the tonnage which can be treated in a given furnace, to 4% S, is quite different from the tonnage the same furnace would treat to 9%, or 12% S. Consequently, not only will the furnace labour and repairs and power cost be less per ton of ore, but also the fuel cost per ton ore,—because only a fraction of the fuel is used for heating the ore,—the bulk of the heat is consumed in keeping the furnace itself hot. For instance, practically the same amount of oil was used when only feeding 20 tons a day, as when feeding 40 tons, and though of course the calcines were not up to specification in the latter case, still the temperatures and hearth atmosphere analysis were.

Just what the chemical reactions were, when the most successful results obtained, it is extremely difficult to state. In order to secure very low sulphur calcines, and to economise on fuel at the same time, free oxidation of the last few units of S, to SO_2 is indulged towards the end, and this SO_2 promptly, and thoroughly reduced to elemental S, as it ascended from hearth to hearth, through strongly reducing atmospheres. The period

of testing was very brief, and sadly interrupted by entirely foreign influences, such as violent storms, electric power breakdowns, etc., and the operations were conducted under demands of almost feverish haste for results before a certain day, so that under the circumstances, the writer feels he may be excused for not attempting too fine an analysis of the strictly technical features. The important fact today is, that in the face of this impossibility for real experimenting, and in spite of the almost blind manner in which many conditions had to be assumed and tried, notwithstanding obstacles that, if foreseen, would have caused many people to refuse to submit an important problem to so great a risk of failure through lack of time, results were actually secured that have claimed attention of concerns whose practice lends itself to the process—practical results indicating a solution of the fume question in many places, and a promising sulphur manufacturing cost. Complete developments had just been arranged for, and new tests were just about to be arranged under most favourable opportunities, in Europe, when the war broke out, and put an effectual stop to all plans.

When it is realized that sulphur, to the value of some $50,000 a day, is being irrevocably wasted from our American roasters alone, and that besides this waste, much damage has been done and is being done to forest and agricultural and ranch lands, it seems strange that more earnest efforts have not been made to conserve some of this value and spare some of this damage. In Tennessee, a great work has been done. There, for years, fumes were emitted, which represented an actual value of $4.50 per ton of ore smelted, the while $4.00 worth of copper was being saved. Not only this, but hundreds of thousands of dollars were spent in law suits and all kinds of trouble growing out of the damage resulting from the SO_2. Now, through steadfast and expensive effort, over $2.00 out of this $4.00 sulphur waste is being saved, and the former trouble-breeding waste product, converted into over 200,000 tons a year of sulphuric acid, is the prime source of profit to the Company. The same waste and the same SO_2 trouble, go on today at many smelting plants. Money is often freely expended to buy land for the sake of the right to destroy it,

money that might have been used for the somewhat better purpose of developing means to stop the nuisance, and conserve the value. If carried successfully into a very general use, the actual profit to be derived from the sulphur produced by this process, would needs be figured at a very conservative sales price, because the recoverable tonnage is so large. But the margin indicated is also large, in many places; so large in fact, that were the market price for sulphur depressed from $20 as at present to $10, the sulphur content of certain ores, and concentrates, which now is a positive expense per ton of ore, would become a positive profit per ton of ore. The $50,000 a day worth of sulphur, noted above, as now being driven off as waste from roasting operations in America, was not mentioned in the sense of a possible recovery, a prize capable of being won, but only as indicating an opportunity in which conservation could unquestionably be cultivated. Both in Europe and Australia the same opportunity exists. This process is one of the first to point clearly to success, by actual full size practice, and confidence is felt that before long all the minor features, which are as yet incompletely worked out, will be fully developed, and this process find its own particular niche in present day metallurgical practice.

THE SMELTING OF TITANIFEROUS IRON ORES

By Bradley Stoughton, New York

Annual Meeting, Toronto, 1915

My excuse for bringing this oft-discussed subject before your attention is, first, because the year 1914 has seen a decided advance in our knowledge of the subject; second, because the titaniferous iron ores of Canada are known to be very vast in extent and not much used, notwithstanding that several attempts have been made in the past to work the deposits commercially; and, third, because it has never been discussed actively at one of the Institute's meetings.

The prejudice which once existed against titaniferous iron ores has now been largely dissipated, or at least has been reduced to an opinion unfavourable to the ores only when certain grades of pig iron, containing more than 1.50 per cent of silicon, are to be made, and when the titanium in the ore is large in proportion. Even then the high-titanium ores may be safely used in admixture with others of a similar kind, and then there need be no limit to the grade of iron made. The truth of these assertions is contained in some smelting results disclosed within the past six months.

Previous Experiences in Smelting Titaniferous Ores

This subject has been extensively studied by such authorities as [1]Riley, Bowron, Rossi, Buller, Buck, and others while electric smelting has been investigated by Lodyguine, Gin, Huppertz, and Stansfield.[2] It has been shown that ores con-

[1]E. Riley, Jour. Chem. Soc., Vol. XVI, 1863, p. 387.
Wm. M. Bowron, Trans. Am. Inst. Min. Engs., Vol. XI, 1882, pp. 159-164.
Auguste J. Rossi, Jour. Am. Chem. Soc., Vol. XII, 1890, pp. 91-117.
Trans. A. Inst. Min. Engrs., Vol. XXI, 1893, pp. 832-867; Bibl. XXXIII, 1903, pp. 179-197.
Charles A. Buck, Iron Age, Vol. LXXXIV, 1909, p. 1223.
W. P. Buller, Iron Age, Vol. LXXI, 1903, pp. 31-2.
[2]A. Lodyguine, Trans. Am. El. Chem. Soc., Vol. VII, 1905, pp. 157-165.
Gustave Gin, Trans., Am. El. Chem. Soc., Vol. XI, 1907, pp. 291-294.
W. Huppertz, Metallurgie, Vol. 1, pp. 362-6, 382-5, 404-417, 458-462, 491-504.
See also Bibliographies in Annals of the N.Y. Acad. of Sci. Vol. I, 1879, p. 53 and Trans. Am. Inst. Min. Engs., Vol. XLIV, 1912, pp. 306-312.
J. W. Evans, Trans. Can. Min. Inst., Vol. XV, 1912, pp. 123-8.

taining from 5 to 40 per cent, of titanium oxide have been smelted with success, notably in Sweden, England, New Zealand, New York State, the Lake Superior District, the Bethlehem Steel Co., etc. In the ores smelted at Norton, England, and in the Adirondacks of New York State, the proportion of[5] titanium oxide was as high as 39 and 20 per cent respectively, while the resulting slags analysed as follows:—

Slag at Norton, England		Slag at Sandford Hill
%		%
27.83	SiO_2	26.72
36.18	TiO_2	25.11
9.18	Al_2O_3	11.86
24.36	CaO	25.81
0.60	MgO	5.99
1.86	FeO	3.46
0	CuO	1.96
100.01		100.91

Smelting was carried on successfully at both of these places, and with no apparent difficulties such as refractory nature of the charge, formation of accretions in the furnace, infusibility of the slags, or excessive amount of fuel required. The fuel charge of the Norton furnace was 17 cwt. of coke per ton of iron, which is not large considering the leanness of the ore—36 per cent iron. It will surprise many metallurgists to be told that slags containing as much as 25 and 36 per cent of titanium oxide are fusible at blast furnace temperatures, but Rossi has shown that combinations of silica, titanium oxide, lime, etc., resembling certain natural minerals, such as sphene, are not difficultly-fusible mixtures, and that, furthermore, titanium oxide and alumina have a fluxing effect upon one another which enables fusible slags to be produced.

But, eliminating the prejudice against titaniferous ores on the score of infusible slags and excessive fuel required, there still remained in the minds of iron masters the objections that titaniferous ores were liable to form infusible accretions in the furnace, which caused blocking of the hearth or hanging of the charge, and that the ores did not always smelt well in admixture with other ores. Against the first of these objec-

tions it was urged that running the blast furnace at a relatively low temperature, so that titanium oxide was not reduced, would avoid the formation of the compounds which caused the accretions. In answer to the second objection it was said that the formation of a slag which would absorb and retain the titanium oxide would enable the titaniferous ores to mix without difficulty. Experiments made in the year 1914, which are about to be described here, have gone a long way towards establishing the complete validity of these arguments, and thus removing the last obstacles to the employment of the titaniferous ores. These experiments were carried on so large a scale, for so long a period of time, and with such perfect frankness and publicity, that the conclusion to which they point will be received by all with confidence.

RECENT SMELTING EXPERIMENTS

In February, 1914, began the most extensive test of the smelting of titaniferous iron ore ever attempted as an experimental operation, and the result of this test justifies the conclusion of Rossi and others, at least to this extent: that the smelting of ore mixtures containing up to 4 per cent of titanium oxide does not necessarily involve difficulties greater than will be met with in the smelting of similar ores free from titanium except only, perhaps, a reduction in the amount of silicon present in the iron produced. Those expected and much-heralded operating difficulties of increased fuel consumption, sticky or infusible slags, salamanders and furnace accretions, and slipping or hanging of the charge, were definitely proved to be bug-a-boos which should no longer frighten the blast-furnace operator away from the use of titaniferous ores. In other words, we may now say that those vast resources of iron ores, containing from 5 to 20 per cent of titanium oxide, may be used without prejudice, either alone or in ore mixtures, in proportions which involve an average amount of titanium oxide in the mixture not greater than 4 per cent. Even this would mean adding at least 20 to 30 per cent of this country's resources of iron ores. But the experiment of 1914 went further than this: it not only proved that mixtures containing as much

as 4 per cent of titanium could be used without detriment, but it gave no indication that mixtures containing more than this would have a bad effect on the smelting operation, either practical or financial. Titaniferous ore was used in proportions up to five-sixteenths of the mixture; delays and difficulties in obtaining more ore of the same kind were the only considerations which prevented greater proportions being used.

Titaniferous Ore Used in the Experiment.—The titaniferous ore referred to above was taken from the famous Sandford Hill magnetite deposit, said to contain many millions of tons in sight, analysing 47.50 per cent of metallic iron, 12.60 per cent of titanium, and less than 0.02 per cent of phosphorus. After magnetic concentration, the ore contained about 55 per cent of metallic iron, 13 per cent of titanium oxide, 0.004 per cent of phosphorus, 0.5 per cent of vanadium oxide, and 0.2 per cent of chromic oxide.[1]

It will be interesting to observe that this is the ore whose successful smelting (without previous concentration) was carried on for many years in charcoal furnaces in the middle of the last century, as described by Rossi, who gives illustrations and diagrams of the furnaces used.[2] In the experiments of 1914 it was planned to use about 30,000 tons of this ore, but the crushing plant which had been erected at the mine proved inadequate for the work; arrangements had to be made for concentrating the ore elsewhere; this involved discontinuance of the use of titaniferous ore from February 16 to June 3, with the result that only about 8,000 tons of this ore were smelted altogether, because the furnace lease expired on July 31.

Blast Furnace used for the Test.—The Port Henry furnace of the Northern Iron Company was leased from Jan. 1 to June 30th, 1914, for the purpose of this test. Owing to the delays we have mentioned, the lease was subsequently extended to July 31, and the furnace was blown out Aug. 3. The experi-

[1]Full details of analysis, etc., will be found in the Year Book of the American Iron and Steel Institute (not yet published); in the *Iron Age* of Oct. 22, 1914; and in the *Iron Trade Review* of Oct. 15, and Dec. 3, 1914.

[2]Trans. Am. Inst. Min. Engs., Vol. XXI, 1892-3, pp. 835-8, 845-6.

ment was under the direction F. E. Bachman, and Geo. L. Shook acted as blast furnace superintendent. The production of the furnace was between 200 and 250 tons of iron per 24 hours.

First Run on Titaniferous Concentrates.—The experiment was started on Feb. 3. Titaniferous concentrates were used to the extent of one-sixteenth of the ore mixture. On Feb. 6, the proportion was increased to one-eight titaniferous concentrates, which mixture was maintained until, on Feb. 16, the titaniferous ore ran out and the furnace was run thereafter on the regular charge of Port Henry magnetic ore. During this run of one-eighth titaniferous ore, foundry iron was made, averaging between 2 and 3 per cent of silicon, 0.015 to 0.035 per cent of sulphur, about .063 per cent of phosphorus, 0.45 per cent of manganese, 0.49 to 0.59 per cent of titanium, 3.45 per cent of graphite, and 0.23 to 0.51 per cent of combined carbon. (See Table II). The slags ran between 30 and 33 per cent of silica, 13 to 15 per cent of alumina and iron oxide together, 2.70 to 4.65 per cent of titanium oxide, 1.40 to 2.05 per cent of sulphur, about 40 per cent of lime and about 9 per cent of magnesia. The only difficulty experienced was an excessively fluid slag, confirming the oft-repeated assertions of Rossi on this point. Otherwise the operations were normal in every respect, and, when the experiment was resumed on June 3, the proportion of titaniferous concentrates used was one-eight from the start.

Second run on Titaniferous Concentrates.—In the second run on titaniferous concentrates, malleable iron only was made, as shown by the furnace record in Table I. The reason for this was that, with 25 per cent or more of titaniferous ore in the mixture, it proved difficult to make iron with more than 2 per cent of silicon, because titanium replaced silicon in the metal. The iron made during this second run contained from 0.70 to 1.70 per cent of silicon, 0.18 to 0.63 per cent of titanium, and 0.015 to 0.041 per cent of vanadium. The slags contained from 25 to 34 per cent of silica, 10 to 15 per cent of alumina and iron oxide together, 2.58 to 10.21 per cent of titanium oxide, and 1.45 to 1.85 per cent of sulphur. Some details are

given in Table II. The slags were very fluid, and it was observed that sulphur could be higher in a titaniferous slag without as much of it going into the iron than was the case with non-titaniferous slags. The experimenters attributed this to the belief that titanium was not so strong an acid in this respect as silicon, but I venture the opinion that an explanation may be found in the affinity of titanium for sulphur, which has been noted by those who have experimented on the treatment of pig iron with titanium.[1]

Indeed, we are told by Dr. Forbes[2] that it was common practice in Sweden in 1874 to add 10 per cent of titaniferous ores to the charge with the view to reduce sulphur.

An increase in the proportion of magnesia increased the fluidity of the slag, but, at the same time, tended to increase the sulphur in the iron. In general the titaniferous slags, although more fluid, melted at a temperature about 100 degrees Fahr. higher than the non-titaniferous slags. This caused the titaniferous slags to become fluid at a lower point in the furnace than the others, and, until this condition was recognized, the slag showed a tendency to hang at the tuyeres and there was an increase in the amount of air blown per pound of coke burned. This indicated to F. E. Bachman that more coke reached the tuyeres and less was consumed by direct reduction of solid carbon in the upper levels.

Difficulties which had been expected in the hanging and slipping of the charge did not materialize; indeed, there seemed to be less tendency to this action when smelting the titaniferous mixture. An examination of the furnace after the run also failed to disclose infusible accretions and salamanders which had been prophesied by certain operators who had had experience in smelting titaniferous ores. Altogether the run was described by the experimenters as satisfactory and extraordinarily free from difficulties.

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[1]See Goldschmidt, El. Chem. and Met. Indus., Vol. III, 1905, pp. 226-227, and Treuhit, Stahl und Eisen, Vol. XXX, July 13, 1910, pp. 1192-1201.

[2]Jour. Iron & Steel Inst., 1874, No. 1, p. 132.

Quality of the Iron Smelted from the Titaniferous Mixture.—The iron smelted from the titaniferous mixtures was tested for strength both alone and in mixtures with other irons. While the results showed no large difference between the titaniferous and non-titaniferous irons, the advantage was with the titaniferous metal. Full details of the tests are given in the references to this experiment cited above.

TABLE I

FURNACE RECORD

Dates, 1914	Proportion of Titaniferous Ore used	Kind of Iron made	Yield of Ore Mixture in Iron Per Cent	Coke—lbs. per Ton of Iron	Limestone		Blast—cubic feet per Ton of Fuel	Furnace Production Tons
					Lbs. per Ton of Iron	Per Cent of Ore Charge		
May 1-8	None	Foundry	64.13	2,269	772	22.1	55.6	1,602
" 8-15	"	"	63.13	2,236	807	22.7	53.7	1,636
" 15-22	"	"	63.15	2,328	964	27.2	55.1	1,611
" 22-31	"	Malleable	60.06	2,781	1,376	36.9	58.8	1,881
June 1-2	"	"	56.78	2,260	1,156	29.3	62.5	1,511
" 3-8	$\frac{1}{8}$—$\frac{3}{16}$	"	59.87	2,403	1,137	30.4	60.6	1,415
" 9-15	$\frac{1}{4}$	"	55.55	2,255	987	24.4	63.0	1,369
" 15-22	$\frac{1}{4}$	"	57.52	2,365	942	24.2	63.2	1,688
" 22-30	$\frac{5}{16}$	"	59.16	2,246	810	21.3	61.8	1,466
July 1-8	"	"	57.34	2,354	865	22.1	66.3	1,287
" 8-15	"	"	63.40	2,453	1,017	28.7	57.4	1,383
" 15-21	"	"	66.29	2,272	892	26.4	62.0	1,745
" 21-31	"	"	.....	.....	....	....	....	....

TABLE II

ANALYSIS DURING TITANIFEROUS SMELTING

RUN	MIXTURE	PIG IRON				SLAGS					
		Si %	Ti %	S. %	P. %	SiO_2 %	TiO_2 %	Al_2O_3 Fe_2O_3 %	S.	CaO	MgO
First Run.....	$\frac{1}{8}$ Ti. Ore	2 to 3	0.49 to 0.59	0.015 to 0.055	0.45	30 to 33	2.70 to 4.65	13 to 15	1.40 to 2.05	40	9
Second Run....	$\frac{1}{8}$ Ti. Ore	0.80 to 1.10	0.18 to 0.47	0.023 to 0.027	0.17	31 to 34	2.6 to 4.7	10 to 11	1.6 to 1.9	.....	.....
Second Run....	$\frac{1}{4}$ Ti. Ore	0.95 to 1.35	0.33 to 0.63	0.015 to 0.029	0.20	30 to 34	5.7 to 7.1	11 to 14	1.7 to 1.8	.....	.....
Second Run....	$\frac{5}{16}$ Ti. Ore	0.70 to 1.70	0.34 to 0.49	0.030 to 0.041	0.14	25 to 30	9.0 to 10.2	12 to 15	1.5 to 1.7	.....	.....

DISCUSSION

MR. F. A. JORDAN.—Mr. Stoughton's information on this subject is of very great interest and importance. If the problem of successfully smelting titaniferous ores in the blast furnace has now been solved, the next step, in this country, at any rate, is to combat the conservatism of blast furnace operators, satisfy them of the practicability of smelting such ores, and thereby conduce to the utilization of Canadian titaniferous deposits. That may be no easy task. According to the figures submitted by Mr. Stoughton the coke consumption per ton of ore smelted is considerably lower than in the smelting of mesabi ore. It would also appear that titaniferous ores can be more cheaply smelted than mesabi hematite.

DR. W. L. GOODWIN.—May I ask Mr. Stoughton what percentage of titanium was contained in the ores to which he has referred?

MR. BRADLEY STOUGHTON.—The ore smelted in England contained 36 per cent of titaniferous oxide. This is a matter of record covering a period of years.

DR. GOODWIN.—It is a well-known fact, there are large bodies of titaniferous ores in Canada, which, however, carry a lower percentage of titanium than the quantity specified by Mr. Stoughton. If the experiment demonstrates the feasibility of smelting these ores, either by themselves or in conjunction with other ores, the sooner we can convince the blast furnace men to that effect the better for Canada.

MR. G. C. MACKENZIE.—We are discussing a very difficult problem. Most furnace-men in Canada are working on hematite ores, and are prejudiced against the use of magnetite, not to mention titaniferous magnetite. I do not believe that these men would entertain the suggestion of smelting ores of the latter class for a moment. I have had the opportunity lately of discussing this subject with certain Canadian furnace-men, and they were most emphatic in declaring that they would give titaniferous ores very little consideration. The experiments at Fort Henry were made by Mr. Bachman, who has had a life-

long experience in the smelting of magnetic ores, more especially fine concentrated magnetites. He has used a furnace specially designed to treat these ores, and was, therefore, in a position to attack the smelting of titaniferous magnetite with some certainty of success. I do not know if Mr. Bachman has had any experience in the smelting of mixtures of hematite and magnetite. Canadian furnace-men would have to consider the smelting of mixtures of hematite and magnetite. I venture to say that if you offered the Steel Company of Canada at Hamilton several thousand tons of titaniferous magnetite, at an attractive price—based on iron content— you could not persuade them to take five tons. If you offered them a magnetite containing no titanic acid, you would have difficulty in selling it. I have yet to discover the Canadian furnace-men who will consider using a magnetic ore, when he can secure a United States hematite ore, at equal prices for equal iron contents.

Another point which needs a little explanation is the concentration of these titaniferous ores. A year or two ago we experienced in Ottawa with a three-ton sample of titaniferous magnetite from Quebec. This ore contained 25 per cent of TiO_2. We crushed the ore so that 50 per cent would pass a 200 mesh screen, and, after wet magnetic separation, found that the concentrate still contained 19.8 per cent TiO_2. This concentrate was re-crushed so that 25 per cent passed a 150 mesh screen, and 75 per cent passed a 200 mesh screen. Reconcentration resulted in a concentrate containing 14.8 per cent TiO_2. The final concentrate contained but 25 per cent of the original iron and 9 per cent of the original titanic acid. It is hardly necessary to say that there would be profit in milling an ore of that description, and, before assuming that any titaniferous magnetite can be concentrated to a low TiO_2 product, it would be well to examine its mineralogical composition. The use of titaniferous magnetites in Canada requires, above all, the education of our furnace-men, and the only way that may be accomplished is to secure the services of Mr. Bachman to build and operate a furnace in Canada. Actual demonstration of the practicability of smelting our titaniferous ores is required before you can convince the present operators of Canadian furnaces.

MR. JORDAN.—Is there not some difference of opinion concerning the success of the Fort Henry experiments?

MR. STOUGHTON.—I was not aware of any difference of opinion. The results have been discussed very widely, and many witnessed the test at the time it was conducted. As a result of the test good iron was produced. That fact is certain. It is true that some of those concerned were rather timid of going further and of using more than five-sixteenths of titaniferous ore; because it was feared that infusible salamanders would form. But when the furnace was blown out no salamanders were found. I should like to add here that the question of the smelting of titaniferous ores has to me merely an academic interest, since I do not own, nor do I expect to own a titaniferous deposit.

MR. MACKENZIE.—There is no doubt at all that Mr. Bachman handled his furnace and surmounted his difficulties. During the latter end of the run he experienced no more trouble with his titaniferous mixtures than many of our furnace-men do with hematite mixtures. It is merely a question of educating our furnace-men to adopt the practice.

PROF. R. H. RICHARDS.—As I understand it there are two requisites for the success of the operation. (1) The fuel must be kept down so as to prevent the heat getting so high as to form freely the cyano-nitride of titanium, which forms the obstructions to the furnace; and (2) the slag-forming elements must be computed so as to leave space with the SiO_2 and Al_2O_3 for the titanium oxide to go into the slag as an acid.

There is one question I should like to ask. Blast furnace slags usually have from 45 to 50 per cent of calcium and magnesium oxides. Mr. Stoughton has made no mention of the percentages in this case.

MR. STOUGHTON.—The slag contained about 40 or 50 per cent of calcium oxide.

MR. ARTHUR A. COLE.—I would ask Mr. Mackenzie if he is familiar with the work done at Radnor in the Province of Quebec.

MR. MACKENZIE.—Mr. John J. Drummond gave me some information respecting this experiment, which was conducted at Radnor some years ago. They used common rock salt as flux, and found that a titaniferous mixture gave no trouble when using a certain proportion of this flux. I have forgotten what proportion of salt was used, or how much TiO_2 there was in the ore mixture. The experiment was carried out by adding small proportions of titaniferous ores, gradually increasing the amount, at the same time increasing the amount of salt as the fluidity of the slag indicated the necessity. The iron made, if my memory is correct, did not contain any titanium, but it was extremely tough, and was reported on most favourably by a well-known English firm. The cost of the salt flux being prohibitive, no appreciable amount of iron was manufactured.

PROF. RICHARDS.—A friend of mine recently visited a plant in England to learn how titaniferous ores were smelted. He found that the furnace in which these ores had been smelted successfully was closed down. On enquiry he ascertained that the success of the operations had been due to the practice of charging enormous amounts of slag in order to flush out the cyanonitride of titanium obstructions. It seems to me, therefore, that it is pertinent to ask if a large amount of slag was used in the experiment.

MR. STOUGHTON.—There was an excessive amount of slag in the ore.

MR. J. W. EVANS.—We began a series of experiments in the direct smelting of titaniferous ores into steel in the electric furnace in Belleville, and also at Montreal, and we obtained some good results on a small scale, and made some very fine tool steel. We then built a 1-ton furnace at Belleville and got out a quantity of tool steel; but had some trouble in the manipulation of the furnace and charge, as a result of not having with us an experienced steel metallurgist. The financial depression affected the people who were associated with us, and we were not able to build a 3-ton furnace as we had expected. The results obtained so far have shown that it is quite possible to make a good quality of tool steel in an electric furnace from titaniferous ore without first making pig iron, and that by using

a smaller amount of charcoal or coke than would be used in a furnace for making pig iron, steel could be obtained in one operation. The burden is made up to contain enough charcoal to ensure the proper reduction of the ore, enough being added to give from 1% to 1.25% of carbon in the steel. It will take sometime to work out a practice on a large scale (say in a 10-ton furnace), but the cost for tool steel would not be more than 5 cents per pound in a 3-ton furnace. It cost us 7 cents per pound in the 1-ton furnace which we had in operation for a short time. We had the steel tested in the C. P. R. shops at North Bay, and it compared very favourably with high speed steel. For working cast iron and .9% carbon steel, it was equal to high speed steel; but did not "stand up" so well when working soft machine steel at a greater speed. The ore we use contained 50% iron, 7.5% titanium, .11% vanadium, .34% nickel, .004% phosphorous, and a trace of sulphur, together with some chromium and manganese.

MILLER'S CHLORINE PROCESS AT THE ROYAL MINT, OTTAWA

By Ralph Pearson, Ottawa, Ont.

Annual Meeting, Toronto, March, 1915

The first use of chlorine gas for the purpose of refining alloyed gold appears to have been made by Lewis Thompson and dates back to the year 1838, when we are told he was presented by the Society of Arts of London with the sum of twenty guineas, and his paper on the subject was published in their transactions. In 1867, F. B. Miller, Assayer to the Sydney (Australia) branch of the Royal Mint, obtained a patent for an "Improved method of toughening brittle gold, of refining alloyed gold and of separating therefrom, any silver they may contain." I think from my knowledge of Miller's contemporaries, that there is no question that he arrived at his conclusions and obtained all his data without knowledge of Thompson's work thirty years before. And I think he is undoubtedly the one to whom credit is due for first applying the process on an industrial scale.

The Chlorine Process is one peculiarly well adapted to the rapid refining of what is called rough gold, that is gold alloyed with varying proportions of silver and base metals. The base metals include everything in the metallurgical calendar, and the proportion varies from 99 per cent gold, 1 per cent silver and base, to 1 per cent gold and 99 per cent silver and base. The average composition in Victoria, Australia, is 91.42 per cent gold, 4.34 per cent silver, 4.24 per cent. base The composition of 108,000 ounces Yukon gold, received at the Ottawa Mint during 1913, was gold 76.77 per cent, silver 19.8 per cent, and base 3.43 per cent.

The amount of base metal contained in a rough gold ingot largely depends on the care and skill of the melter at the mine where the gold is produced. Two neighbouring mines working on ore of approximately the same composition will produce ingots varying widely in their contents of base metal. The one has possibly 75 per cent gold and silver, and 25 per cent base, the base consisting of 15 per cent lead and 10 per cent copper and arsenic. For the removal of large quantities of base, the Mint charges an additional fee. The other melter by the simple

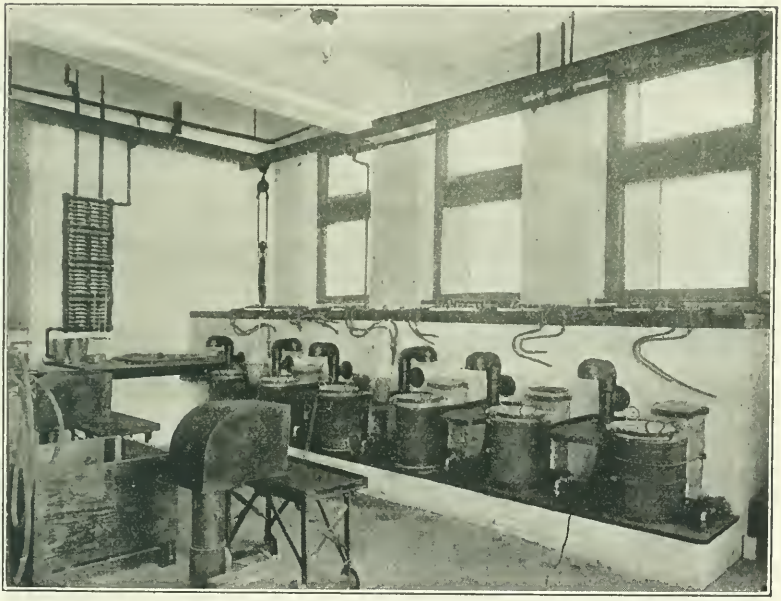
use of nitre or any other convenient means, sends out an ingot containing say five per cent base, this saving the mine owners many dollars in charges for additional treatment before a reliable assay is obtainable.

The refinery at the Ottawa branch of the Royal Mint began work in January, 1911, the equipment being entirely for the electrolytic deposition of gold and silver. The capacity of this equipment was soon found to be quite unequal to the supply of rough gold sent from all parts of the Dominion and beyond. The Government was at first informed that no Yukon gold would be sent to Ottawa, and yet the refinery has treated Yukon gold alone to the value of $5\frac{1}{4}$ million dollars. The Porcupine district introduced a new contributor; the gold of Ontario origin received at the Mint in 1912 had a value of $196,000, and in 1913, over $1\frac{1}{2}$ millions, it was therefore decided that, as the first part of the electrolytic process, viz: the plating out of the silver contents required a very considerable part of the time available, it would be advisable to replace this by the Miller Chlorine process, and the existing building was extended to accommodate the equipment for this.

The equipment for the Miller process is comparatively simple and is as follows:—A constant supply of chlorine gas at a pressure of not less than five pounds to six pounds per square inch, melting furnaces either coke or oil, condensing chambers at rear of furnaces, and suitable pots, tongs and moulds.

At the Ottawa Mint, the chlorine gas is delivered to the furnaces from steel cylinders, each holding 100 pounds liquid chlorine at a pressure of 97 pounds per square inch, at 20° C. The cylinders are in a brick cupboard having a tightly fitting door and ventilating pipe passing to the furnace flue, and also a pipe communicating with the outside air, so that there be a constant circulation and any accidental escape from the cylinders would not find its way into the building. Two cylinders are used, connected by a common header to duplicate half inch, nine pound lead pipes, and so arranged that either cylinder may be used on either pipe line by simply opening and closing the necessary valves. The lead pipes are carried to points immediately above the furnaces where glass stop cocks are inserted for regulating the flow of gas and are connected with the lead by

heavy rubber tubing. Two branch pipes are connected with a lead receptacle having three necks similar to a Woolff's bottle, the third neck holding a lead pipe connected to a glass tube which in turn is attached to a lead bulb, very similar to an enlarged pipette. The lead vessel holds sufficient water to give a column eleven feet in height, corresponding to a working pressure of about five pounds per square inch. Should the pressure exceed this owing to suddenly shutting off the supply to several furnaces, then the water is forced up into the lead bulb and the gas escapes outside the building, thus preventing any danger of bursting the rubber connections or blowing the liquid metal out of the crucible.



Furnaces, Chlorine Pots and Pipe Stems. In front Hooks connected to Fan and overlapping Iron Pouring Tables.

The furnaces are fire clay cylinders 18 inches high, 14 inches diameter, encased in sheet iron and having an inlet at the bottom for the atomized oil and air, and a small outlet, $3\frac{1}{2}$ inches in length, 2 inches wide, placed two inches below the top for escape of products of combustion, etc., the interior is cylindrical 15 inches deep, $8\frac{1}{2}$ inches diameter, the furnace covers are fire clay

Washing Tower and Pipe to Exhaust Fan in Basement.

made in two parts having a one inch hole in centre to allow the passage of the earthenware chlorine pipe. These furnaces burn crude fuel oil, 86 Sp. gr., at a pressure of 40 to 50 pounds per square inch and air at two pounds per square inch. The flame

enters at a tangent and whirls around in the form of a spiral, finally reaching the outlet at the top, which is connected by a short flue, nine inches long with the condensing chamber 5′ x 4′ x 20′ in length. This in turn is connected with a similar chamber running at right angles to the first one, also 20 feet long but three feet six inches wide by three feet high. From the end of this chamber the flue is carried to the foot of a washing tower, 20 feet high and 30 inches diameter, and having a water spray attached to the centre of the top-piece. This tower is connected with a 12 inch earthenware exhaust-fan, driven by a $7\frac{1}{2}$ h.p. motor at 1,000 R.P.M. propelling 4,400 cubic feet of air per minute. The bottom section of the tower stands in an earthenware tank to collect any material washed from the fumes by the water spray. The fumes resulting from the chlorination and all furnace gases are thus immediately drawn from the furnaces through the chambers, and pass up the washing tower against a spray of water before being blown out of the chimney.

The crucibles are known as 4-pint chlorine pots and are made of clay, the outside measurements being, height $11\frac{1}{4}$ inches; diameter, $5\frac{3}{4}$ inches; tapering at the bottom to $3\frac{1}{4}$ inches; walls, $\frac{3}{8}$ inch. Owing to the fact that fused silver chloride will pass through a clay crucible almost as rapidly as through a sieve, they are thoroughly soaked in a saturated solution of borax before being used. After this treatment they will withstand the action at a red heat of silver chloride and lead chloride for several hours without undue injury.

500 to 700 ounces of the metal to be refined (together with sufficient borax to form a layer $\frac{3}{4}$ inch deep on the fused metals), are melted in the clay crucible standing in a plumbago guard pot to catch any leakage due to a defective crucible or an accident. The clay crucible is covered with a clay lid having a slot for the passage of the clay pipe stem. These stems are made of white clay and are 25 inches long, $\frac{1}{2}$ inch diameter, $\frac{1}{8}$ inch bore, and connect with end of lead supply pipe by a flexible rubber joint. When the metal and borax are melted the pipe stem previously heated to redness is introduced through the slot in the lid, pushed to the bottom of the crucible and held in position by a clamp. The chlorine is slowly admitted by cautiously opening the glass stop cock controlling the supply to that particular furnace, until

the gas is passing through the metal with a steady pulsation. As lead and copper are nearly always present, dense fumes of the chlorides of these metals immediately appear and practically the whole of the chlorine is absorbed. As the action continues, silver chloride is formed and floats on the top of the gold together with some fused copper chloride underneath the layer of borax. During the process all base metals are converted into chlorides and expelled, finally leaving the gold, the fused silver chloride, and the borax cover. The finishing point is judged by the brown stain formed on a piece of cold pipe stem held in the fumes issuing from the pot; and workmen become very expert in judging the finishing point by the colour of the flame. When it is considered that the chlorination is complete the pot is withdrawn from the furnace and placed under a hood, (connected with the flue), for a few minutes until the gold is set. The silver chloride and borax are still liquid and these are poured off into iron moulds placed under a hood connected with the exhaust fan. The moulds are 12 inches by 12 inches, by 2 inches deep. The pot is now inverted on an iron table when the cone of fine gold falls out. This is quenched, cleaned and several melted together into ingots of convenient size, the average assay being 995.0. Gold refined in this manner is quite tough, the difference between 995.0 and 1,000 being silver and a little copper. If gold of a higher fineness is required, it is easily and rapidly produced by the usual Wohlwill electrolytic process, the chlorinated gold being used for the anodes. By this means the fineness may be raised to 999.7—999.9.

When the work is done entirely by electrolytic methods, the anodes for the Wolhwill process produced from 'Black Gold,' i.e. the spongy alloy left after the plating out of the silver in a solution of argentic nitrate and nitric acid, contains 94 per cent gold and 6 per cent silver and base, the silver forming large quantities of 'slimes,' and the removal and treatment of these slimes necessitates extra labour. By the use of finer anodes resulting from chlorination the slimes are materially reduced, gold of high degree of fineness is produced and any traces of platinum are still recoverable from the electrolyte.

It may perhaps be thought that the treatment of the slimes mentioned above is offset by the recovery of silver from the fused

silver chloride resulting from chlorination, but such is not the case, the labour and material used in handling it being very much less than is required for an equivalent amount of 'slimes.'

The time required for a chlorination is of course entirely dependent on the amount of metal to be removed; for instance, an ingot weighing 500 ounces and containing 140 ounces base, chiefly lead, and 80 ounces silver, required 7 hours. When a large quantity of silver is present it is necessary to 'dip' the fused silver chloride out with a small crucible, otherwise it would overflow. The silver chloride contains some gold which is removed in the following manner. The cakes are broken up and melted in Number 40 plumbago crucibles, and kept at a temperature a little above the melting point of silver, and an amount of sodium carbonite equal to 10 per cent of the weight of chloride is thrown slowly on to the surface. This produces a shower of silver globules which collect the particles of gold and sink to the bottom. A second addition is made after a few minutes and the resulting shower of globules allowed to settle. It then stands 5 or 10 minutes, and the pot is withdrawn from the furnace, cooled below the melting point of silver, and the still liquid silver chloride poured out. At the bottom of the pot is a large silver button containing all the gold.

The cakes of gold-free silver chloride are now broken into pieces approximately $\frac{1}{2}$ inch square, folded in flannel bags so as to form a parcel about 18 inches long, 2 inches thick and 12 inches wide. Each parcel is placed in a cane basket and a number are piled on top of each other in a tank full of hot water and kept hot by steam. The tank is divided into two compartments admitting free circulation of the liquid; any copper chloride is in the course of 48 hours dissolved out; and as the liquid circulates it comes in contact with scrap iron in the second compartment and is reduced to metallic copper, forming a sludge which settles at the bottom of the tank. When all the copper chloride is removed the silver chloride is emptied out of the bags, washed by decanting, acidified with hydrochloric acid, and reduced to metallic silver by contact with iron plates. The resulting silver is washed and melted in black pots, and cast into ingots, with fineness varying from 995.0 to 998.0, and is practically gold-free.

GOLD ON THE NORTH SASKATCHEWAN RIVER

By J. B. Tyrrell, Toronto, Ontario

Annual Meeting, Toronto, March, 1915

During the past half century the North Saskatchewan River has yielded more than half a million dollars worth of gold to the placer miners who have toiled with shovel and grizzly on its gravel bars, and who, in later years, with larger ambition, have used dredges to dig up more gravel from beneath greater depths of water than it was possible for men to dig with their shovels alone.

Physiography

The North Saskatchewan River rises in a number of rapid streams which are fed by glaciers and snowfields in the high interior ranges of the Rocky Mountains. Before issuing from the mountains many of these streams unite to form the main branch of the river, which then flows eastward and at about 150 miles from its source is joined by the Clearwater river near the site of the old fur-trading post of the Hudson's Bay Company, called Rocky Mountain House. Below the junction of the Clearwater it turns abruptly northward, and after keeping to this direction for forty or fifty miles again swings gradually eastward, which general direction it maintains past the towns of Edmonton, Battleford, Prince Albert, and Le Pas, until it empties into the great north-western expansion of Lake Winnipeg. Its total length in this distance is about 1,250 miles.

In the mountains its many branches flow in steep-sided mountain valleys. After leaving the mountains, the main stream cuts through the wooded ridges of the foot hills and then it flows across the great plains in a valley from 100 to 300 feet deep and a mile or more in width. Finally, 280 miles above its mouth, it leaves this valley and flows in a comparatively shallow channel through a low-lying forested country to Lake Winnipeg.

The general width of the stream in the vicinity of Edmonton is from 400 to 900 feet.

The volume of water varies greatly at different seasons of the year. In the autumn, winter and spring it is very low, but great floods come with the long hot days in summer, when the snow is melting rapidly in the mountains. At such times, when other prairie streams may be almost dry, I have known the water in the North Saskatchewan to rise twenty feet in twenty-four hours with a rush that would sweep away every obstruction in its course.

The average flow of the water in the river at Edmonton varies from 1,500 to 20,000 cubic feet per second, but in extreme high water it has been known to rise to 180,000 cubic feet per second.

History

When gold was discovered on the Fraser River in 1857, glowing accounts of the discovery quickly spread over the whole continent, and people began to rush to the new goldfields from all directions. In the spring of 1858 at least two parties of gold seekers started northward and westward from the State of Minnesota across the plains of the Saskatchewan to the new discoveries. One of these parties crossed the Rocky Mountains in October of the same year, while the other party only managed to reach Fort Pitt, a trading post of the Hudson's Bay Company on the Saskatchewan River above Battleford, where the men remained throughout the winter, and where in a quarrel one partner was shot in cold blood by one of the others. In the following spring these men continued on their way across the mountains to the Fraser River.

In 1859, parties of gold seekers continued to cross the Canadian plains on their way to the diggings on the Fraser River. One of these parties, of whom a record remains, consisted of Messrs. Colville, Reid, Dickman, Hind and two or three others, this party, having been met by the Earl of Southesk, near the Bow River, just as it was about to enter the mountains. During this year also, Dr. James Hector, afterwards Sir James Hector, was exploring on the Saskatchewan, and he states that somewhere in the vicinity of Edmonton "I have seen a few specimens of gold washed out."

In the fall of 1860, or in the spring of 1861, some American prospectors discovered gold, in what was considered to be paying quantities at the Rocky Mountain House, but whether these prospectors were going westward on their way to the Fraser River or to the newly discovered diggings at Cariboo, or had come eastward across the mountains from British Columbia, is not certain.

In May 1863, when Lord Milton and Dr. Cheadle were at Edmonton, they speak of a party of miners coming in from White Mud Creek, about fifty miles farther up the Saskatchewan, where they had been washing gold. The leader of the band was a Kentuckian named Love, who had been in British Columbia searching for gold and had come eastward into what is now the Province of Alberta, through the Yellow Head Pass. He asserted that already that season the miners had recovered £90 apiece.

Another miner mentioned by Milton and Cheadle as being in the district at the time was Perry, who was well known on the Fraser River.

In 1866, reports of rich discoveries of gold on the Saskatchewan River spread among the miners on the Kootenay River, in the vicinity of Wild Horse Creek, and caused a stampede to Edmonton, but as the gold-bearing bars on the river were not found to be as rich as had been reported, the excitement soon died down.

For the next twenty years miners worked on the bars for about 75 miles up and down the river from Edmonton, with results which are said to have varied from one to twenty dollars a day per man.

In 1887, the first dredge was built on the river and from that time on to the present dredges have been working with varying success, though it is probable that most of the gold that has been extracted down to the present time has been taken out by miners working with shovel and grizzly as of old.

The following is a list of some of the references to the occurrence of gold in this river.

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Selwyn, A. R. C.—Observations in the North-West Territory on a Journey, etc. Report of Progress Geol. Sur. Can., 1873-4, p. 58.

Ells, R. W.—Report on the Boring Operations in the North-West Territory, Summer of 1875. Report of Prog. G.S.C. 1875-6, p. 289.

Tyrrell, J. B.—Report on a part of Northern Alberta. Ann. Rep. G.S.C. Vol. 2 (1886), Part E. Ottawa, 1887, pp. 12, 109, 134-5 and 151-2.

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Hoffmann, C.—Chemical Contributions. Ann. Rep. G.S.C., Vol. V. (1890-1), p.p. 66A and 65R.

Records the occurrence of platinum with the gold.

Dawson, G. M.—Ann. Rep. G.S.C. Vol. VIII (1895), pp. 16-7A.

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McLeish, John.—Mineral Productions of Canada, 1912.

Ann. Rep. Dept. of Mines, Ottawa, 1914, p. 69.

Occurrence

The gold occurs in very minute flakes and particles, associated with sand and gravel, from Rocky Mountain House all the way down to Prince Albert, a distance of 650 miles, but it occurs in greatest abundance on that portion of the stream beginning near Goose encampment, where the big coal seam at the top of the Edmonton series crosses the river down past Edmonton to the vicinity of Beaver Lake Creek, a total distance of about 130 miles. Throughout the whole of this distance the coal bearing Edmonton series forms the banks of the valley.

The gold, though in very minute flakes, is found associated with coarse gravel on such bars in the stream as are uncovered at low water. Many people consider that it should be found in the deepest parts of the river, where the bottom is usually

composed of soft mud, and one of the first dredging enterprises was started with the avowed intention of digging up the mud from these deep pools and recovering great quantities of gold, but the enterprise was a failure and those who were in charge of it were forced to learn that the heavy gold had been concentrated by the stream along with the coarser gravel, and that it must be looked for with this gravel and not with the fine and lighter mud.

Many people were also obsessed with the idea that they would find a pay-streak of coarse gold on bedrock, but the coarse gold was not present, and the conditions of the formation of the present river valley, the character of the bedrock, the fineness of the gold, and its perennial supply were all conditions which militated against the formation of a pay-streak on bedrock.[1]

Every year, in times of low water, men foregathered on the gravel bars beside the stream, washed the gravel in grizzlies and collected the gold on blankets, after which the blankets were washed and the gold was separated from the rest of the tailings by means of mercury.

At the present time gravel is being dug by dredges from the bed of the stream for use on the streets of Edmonton and for building purposes in the town. Before it is fit for use in cement it must be washed and it is now customary to wash it in sluice-boxes, and while freeing it from mud to recover any gold that may be associated with it. The gold so obtained is said in some cases to pay for a considerable portion of the expense of excavating the gravel.

A small amount of platinum is associated with the gold on the river, but up to the present time very little effort has been made to save it.

PRODUCTION

The returns of gold from the Saskatchewan River for twenty-six years, from 1887 to 1912, as given by the Department of Mines of Canada, are 14,684 fine ounces, of a value of $303,549.

As gold was discovered in workable quantities on the river in 1861, the time from 1861 to 1886 represents another and earlier

[1] See J. B. Tyrrell "Law of the Pay-streak." Trans. I.M.M. Vol XXI, 1912., pp. 593-613.

period of twenty-six years, during which mining on the river bars was actively prosecuted, and during which these bars, being new and previously undepleted, were undoubtedly richer than they were during the succeeding period, so that it is reasonable to assume at least as large a production for the twenty-six years from 1861 to 1886 as from 1887 to 1912. On this assumption the river would have yielded, up to the end of 1912, gold to the value of $607,098.

In this connection, however, I would like to offer the suggestion that the Government of the Province of Alberta should appoint a Royal Commission to collect evidence from old miners and others as to the actual quantity of gold that was recovered on the river in early times, the positions of the bars that proved to be richest, and any other information that may now be stored away in the memories or note-books of some of the old men, but which will very soon be irrevocably lost.

Geology

The occurrence of gold on the North Saskatchewan is so intimately connected with the character of the rocks which underlie the country, the relationships of these rocks to each other, and their mode of origin, or in other words with the structural and historical geology of the country through which the river flows, that some idea of the progress of geological events in the Rocky Mountains and the country to the east of it is necessary before the extent and mode of occurrence of the gold can be clearly understood.

The Rocky Mountains, in which the Saskatchewan River takes its rise, consist of quartzites, sandstones, conglomerates, shales, and limestones, with many different local characteristics. They outcrop in attitudes varying all the way from horizontal to vertical and with a general strike in a north-west, south-east direction. All these rocks are detrital and consist of sediments which have been deposited in water. Igneous rocks, such as granites, diorites, etc., are almost entirely absent.

After the river issues from the Rocky Mountains its banks are composed chiefly of stratified sandstones and shales with occasional beds of coal. At first these beds are seen to dip at

high angles, but as the river is descended they gradually flatten out until they come to lie so nearly horizontal that the eye cannot detect the difference. In he vicinity of Rocky Mountain House the rocks are thick bedded brown sandstones, while lower down the river these are replaced by brown and white sandstones and sandy clays. The sandy clays prevail, especially for about fifty miles above and below Edmonton, when they in their turn gradually rise to the top of the bank and dark clay shales come up from beneath them and take their place. Thence down the stream to Cumberland Lake, these clay shales, known as the Pierre shales, are present in the banks of the valley.

On the terraces which are present on the sides of the valley east of the mountains, the sandstones and shales are overlain by beds of gravel of early glacial age, which have been called by Mr. McConnell, now the Director of the Geological Survey of Canada, the Saskatchewan Gravels.

Over the sandstones, shales and gravels is a mantle of boulder clay which, in descending the river, is first met with some little distance below Rocky Mountain House as a thin sheet of unassorted clay, or merely as a few erratics scattered over the surface. The sheet of clay becomes gradually thicker as the river is descended, until in its lower portion from Prince Albert downwards, the banks are composed largely of it. This sheet of stoney clay is found on examination to consist of materials derived from the underlying shales and sandstones mixed with boulders and some fine rock flour brought from the Archean peneplain to the north and north-east. These materials have all been kneaded together by the great ice sheets of the Glacial period into one homogeneous, unstratified, stony mass.

In addition to the beds above enumerated, gravel, sand, and mud of recent age occur in the bottom of the stream and on the the low alluvial flats which bound it on one or both sides.

It is from these sands and gravels that the gold has been chiefly recovered, but a certain quantity of gold has also been recovered from the Saskatchewan gravels which lie on the terraces above the level of the river.

The rocks, sands, and gravels which have just been enumerated have been formed under conditions which may be briefly outlined as follows:

In early Pre-Cambrian times a land area, formed of gneiss schist, etc., existed in that portion of British Columbia which lies west of Canoe River and thence southward to the Arrow Lakes, where the gold range at present stands. To the east and west of this land area was an Oceanic Depression, but the western depression does not concern us here, so we will confine ourselves to the consideration of the eastern one, and in order to clarify our ideas and to keep clearly in mind the particular body of water in which the rocks above enumerated were deposited, we will call this eastern Oceanic Depression the *Albertan Sea*.

Throughout the whole of Paleozoic times sand and clay were carried from the land area on the west into this Albertan Sea, and beds of limestone were also formed in it. As the beds of sandstone, shale and limestone gradually became thicker and thicker the floor of the Sea continued to subside, until at the close of the Paleozoic era an immense body of stratified deposits, nearly nine miles in thickness, had accumulated on the Ocean floor along the line now occupied by the Rocky Mountains.

About the beginning of the Mesozoic era the western portion of the area occupied by this great thickness of stratified deposits rose above the level of the ocean, but nevertheless the Albertan Sea, though greatly narrowed by the elevation of its western shore, continued to exist, and drainage channels doubtless continued to flow into it from the Old Gold Range to the west across the newly risen country. Since the rising of this land was doubtless gradual, Mesozoic deposits were laid down near the western shore of the Albertan Sea conformably on the older Paleozoic sandstones, limestones, etc., to the west of them.

This deposition of sediments in the Albertan Sea continued with comparatively little interruption throughout Mesozoic times. About the beginning of the Cretaceous epoch, however, warping and folding began in the Rocky Mountain belt, while the Albertan Sea became gradually enlarged through the sinking of the country to the east, until by Middle Cretaceous times it had reached eastward as far at least as the present Lake Winnipeg, and from that time onward until the close of the Cretaceous Epoch extensive beds of shales and sandstones were deposited on the

floor of this new portion of the sea. Also, until the end of Cretaceous times the main drainage channels of the western country appear to have continued to flow eastward across the gradually rising land to the west of it, carrying the drainage from the sources of these streams down into the Albertan Sea. Some of these streams doubtless flowed from tracts of the country which were rich in gold, while other streams flowed over country which did not contain gold.

Towards the close of the Cretaceous epoch the Albertan Sea gradually began to be cut off from the main ocean and to become shallow, so that the sandstones and clays which were deposited during the terminal period of the Cretaceous, which is known as the Edmonton period, were deposited in shallow brackish water, with the formation of great beds of coal along the coast line of this dwindling sea. During this Edmonton period the streams from the west still took their rise in the mountains of the Gold Range, although they flowed across a belt of land from 100 to 200 miles in width before they discharged their load of sand, accompanied by a little fine gold, into the Albertan Sea.

At the close of Cretaceous times the Laremide revolution set in, when the rocks of the Rocky Mountain belt became intensely folded and great faults developed, along which the older rocks were shoved up and pressed over the newer ones for distances which were sometimes as great as six or seven miles. This mighty revolution and upheaval, during which the Rocky Mountain belt suffered a contraction of about one-half its original width, cut off the drainage which had flowed across it, and from that time onwards the summit range of the Rocky Mountains formed the water shed for the eastern drainage, which for a time continued to flow into the diminished representative of the Albertan Sea, now separated from the ocean and occupied entirely by fresh water. In this great fresh water lake the sandstones, etc., of the Pascapoo or Lower Eocene series were deposited.

During the remainder of the Tertiary epoch the mountains and plains through which the North Saskatchewan flows were suffering active erosion, and in places many hundreds, or even

thousands, of feet of the surface were removed by atmospheric and stream erosion and carried away to the ocean.

At the close of the Tertiary era the Glacial period began. At first glaciers formed on the Rocky Mountains and moved eastward down the main valleys and out towards the plains, while from the faces of these glaciers streams loaded with gravel rushed on down the valleys, and, depositing their loads, formed the Saskatchewan gravels.

At a later time ice sheets developed on what is known as the Keewatin Glacial Centre, far to the north-east on the Archean peneplain, and from that centre extended over the whole country southward and south-westward to within a short distance of the base of the Rocky Mountains, carrying with them material from the Archean and Paleozoic rocks over which they moved, and leaving on their retirement a great sheet of stoney clay or boulder clay spread over the surface. In places, especially on the eastern parts of the plains, the glaciers filled up the old Pre-Glacial valleys with this stony clay, while in other places, as in the country about the upper parts of the North Saskatchewan River, they dipped into the valleys and merely deposited a thin layer of stony clay in these valleys over the surface of the sandstones, shales, and Saskatchewan gravels.

When the Keewatin glaciers had retired, the North Saskatchewan River, like most of the other rivers flowing from the mountains, resumed its old course. In those portions of its valley which had not been reached by the Keewatin glaciers, or in which the Keewatin glaciers had deposited but little stony clay, it continued to deepen and widen its valley as before, and where, farther east, the glaciers has almost blocked its valley with a heavy deposit of clay, it had re-excavated its channel so that it is now everywhere flowing on a bed of the underlying rock.

On this bed of rock, which being shale or clayey sandstone has usually a smooth or slippery surface, recent gravel, sand and mud have been deposited, except on those places where the river is at the time actively engaged in deepening its channel, and with the coarser of these sediments the gold which is being fed into the river from time to time is concentrated.

Source of the Gold

During the early days of mining in the river, many years of time and many thousands of dollars of money were spent in searching in the mountains and on all the upper branches of the river for the lodes from which the gold was supposed to have been derived, but without favourable result.

As early as 1861 Dr. Hector, commenting on the reported discovery of gold in the Saskatchewan, states, "There is no trace on that side of the axis (of the Rocky Mountains) of the ancient rocks from which it must have been derived," but he has no suggestion to offer as to whence it was derived.

When Dr. A. R. C. Selwyn, then Director of the Geological Survey of Canada, descended the river in 1873, he recognized that the gold was not derived from the mountains at the source of the river, and that the search for coarse gold and 'mother lodes' in that direction was hopeless. As a forlorn hope he suggested that it was washed out of the boulder clay which had been transported across the great plains from the Archean peneplain far to the north-east. The problems of glacial transportation were not then understood as they are now, and the drift was supposed to have been transported by water and icebergs, whereas we now know that it was carried by ice moving over the land.

At the present time no one would think of urging that the gold in the Saskatchewan and the other streams flowing from the mountains was derived from the till or boulder clay, knowing that the upper portions of these streams, flowing through parts of the country which had been near the border of the ice-sheet, where the quantity of till derived from the Archean peneplain was very small, were rich in gold, while the lower portions of those same streams, flowing through thick beds of till nearer the source of supply, and consequently richer in such heavy metals as gold, contained no gold. If there had been gold in the boulder-clay it would have been more abundant in it near the source of supply, and where the streams in their lower portions flowed through thick beds of this boulder-clay they would certainly, in case the gold had been derived from the boul-

der-clay, have been much richer in gold than in those portions which lay around the extreme edge of the ice-sheet.

If the gold on the Saskatchewan River had been derived from the boulder-clay which occurs in decreasing quantity on its banks as the stream is ascended, the ice sheet of the Glacial period, by means of which this sheet of boulder-clay was carried to its present position, must have carried all the gold-bearing portion of this boulder-clay over the north-eastern parts of the Great Plains without dropping it, while it dropped most of the rest of its load, even though composed of lighter material, until finally, when it reached its farthest western extension, it sprinkled its load of gold over the country near the foot of the Rocky Mountains.

Such a reversal of ordinary natural laws did not constitute a satisfactory explanation of the source of the gold, so that some other explanation had to be looked for.

In the year 1886, while investigating the geological features of the country along the Saskatchewan river for the Geological Survey of Canada, an old prospector named Haney drew my attention to the fact that gold occurred in what he considered to be a volcanic 'blowout' on the north bank of the river, which 'blowout' proved to be the clinkers and cinders left from the natural burning of the Big Coal Seam over a considerable area. Haney's assays of the cinders yielded small quantities of gold, and the assays of my own samples, not only of the ashes and cinders of the coal, but of the adjoining rock, shewed traces of gold.[2]

A somewhat similar occurrence of gold in association with coal of Cretaceous age, though slightly older than that on the Saskatchewan, has recently been recorded by R. W. Stone, of the United States Geological Survey, from the Cambria Mine in the State of Wyoming.[3]

E. T. Dumble has also recorded the occurrence of gold in lignitic sandstone in the lower Eocene of Texas.[4]

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[2]Report on a part of Northern Alberta, by J. B. Tyrrell. Ann. Rep. G.S.C. 1886. Part E., p. 109.

[3]R. W. Stone. Coal near the Black Hills. Bull. 499. U.S.G.S. 1912. pp. 63-4.

[4]E. T. Dumble. The Occurrence of Gold in the Eocene Deposits of Texas. Trans. A.I.M.E. Vol. 44, 1912., pp. 588-591.

This discovery of gold in association with a bed of coal at the top of the Edmonton series of the Cretaceous provided a clue to a rational explanation of the origin of the gold on the river.

Fuller investigation showed that that portion of the river in which the gold was most abundant was underlain by the coal-bearing rocks of the Edmonton series which were deposited in their present position just before the Laremide revolution raised the Rocky Mountains to their present altitude and grandeur, and that after a particularly heavy flood had swept down the river, and had washed down and re-assorted heavy slides of the soft Edmonton rocks, the bars would be enriched with a new supply of gold. It was also recognized that the rocks of the Edmonton series were not such as might have been derived from the degradation of the quartzite, slates and limestones of the Rocky Mountains, but that the material of which the sandstones especially were composed, "was such as might be expected to be derived from the degradation of granitoid rocks." On this evidence the theory was advanced that the fine gold in the river was derived directly from the Cretaceous (Edmonton) rocks which formed the banks of the stream, and that these rocks in their turn were an old and very low grade placer deposit which had come originally from the mountains west of the Upper Columbia valley.[5]

After the publication of the report of my investigations in 1887 useless and expensive search for the source of the gold in the mountains almost entirely ceased, and the attention of the miners was devoted to the exploration of the bars within the productive area marked by the outcrops of the rocks of the Edmonton series, so that while in 1887 the production had fallen to $2,100.00, in 1896 it had again risen to $55,000.00.

It has been shown that the rock at the summit of the Edmonton series, associated with the Big Coal Seam, contains gold in small but appreciable quantities, but it has not yet been shown whether there is some band of clay or sandstone in this series which is decidedly richer than the others. It is just pos-

[5] J. B. Tyrrell. Report on Northern Alberta. Ann. Rep. G.S.C. 1886. Part E. pp. 134-5 and 151-2.

sible that such a bed might be discovered which would be rich enough to pay for working, or failing this a bed might be discovered and identified which could be assisted to fall into the river, to be there assorted and concentrated, with the purpose of quickly enriching the bars for the miners.

The supply of gold to be obtained from the sandstones which form the banks of the Saskatchewan valley is unlimited, but nature is drawing on this supply very slowly. It is quite possible, however, that nature could be easily assisted in procuring the gold from these sandstones to the great benefit of the mining industry of the Province of Alberta.

A NEW GOLD AREA IN NORTHERN SASKATCHEWAN AND MANITOBA*

By E. L. Bruce, Ottawa, Ont.

Annual Meeting, Toronto, 1915.

The discovery of valuable mineral deposits in basic rocks of Pre-Cambrian age in so many places in Canada has led to the prospecting of even small areas of such rocks, although they may be far removed from any known ore-deposits. It has been known for some time that Pre-Cambrian greenstones and schists occur to the northwest of Lake Winnipeg, around the lakes at the headwaters of the Nelson river, but only recently have they received much attention from prospectors and even yet, although some properties have now reached the development stage, a large part of the district underlain by promising formations is practically untouched.

The part of this belt that has aroused most interest lies around the north end of Amisk or Beaver lake. It is reached most easily from Pas, Manitoba, where the Hudson Bay railway crosses the Saskatchewan river. Boats run from that point to the head of Sturgeon lake, from which a wagon road leads to Amisk lake, sixteen miles to the north.

From Pas north to the middle of Amisk lake the surface consolidated formations are flat-lying magnesian limestones of Silurian and Ordovician age. Emerging from beneath the Ordovician, the older complex forms the surface rock of the northern end of the lake. Usually the limestones have massive beds some distance above the base, underlain by thinner and less resistant beds, with the result that their northern border is a well marked cliff. The trend of this escarpment is roughly east and west. The main mass of greenstones and related rocks extends northward from the escarpment for distances up to twenty-five miles. The western boundary is a short distance west of Amisk lake while eastward the belt extends, with some interruption, a hundred miles to Lake Wekusko.

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The relief of the country is low, the most striking difference in elevation being at the northern edge of the Ordovician sedimentaries where the limestone escarpments overlook the lower, Pre-Cambrian areas. There, the range in elevation is sometimes as much as a hundred feet. In areas underlain by the older rocks the difference is seldom more than forty to fifty feet. In the western part of the area, low, narrow ridges run north and south, separated by narrow, swampy, peat-filled valleys. There are practically no glacial deposits, except an occasional low ridge of boulder clay or gravel, lodged behind a protecting protuberance of rock. The eastern half, however, lies in the basin of glacial Lake Agassiz and the surface is mantled by a thick deposit of fine sediments. As a result, the western half has a great many more small lakes than the eastern, clay-mantled part, although waterways are numerous enough in all parts to make transportation by canoe fairly easy.

The crystalline rocks are of Pre-Cambrian age but being far removed from the type areas, close correlation of the various types cannot be attempted. In character, however, the rocks of the complex in this district are comparable to the Keewatin greenstones and schists, the Temiskaming sedimentary series and the granite intrusives of Lorrain age as described in Ontario Pre-Cambrian areas. Whether they are equivalents of the eastern series or not it is impossible at present to say and as the Northern Saskatchewan area is entirely isolated, correlation may never be absolutely certain. As in most localities, a profound unconformity, representing a long time interval, separates the crystallines from the overlying Palaeozoic limestone.

The oldest rocks of the Pre-Cambrian complex are massive and schistose greenstones and associated green chloritic and grey micaceous schists. Where massive, as they are on many of the islands and along the north shore of Amisk lake, the greenstones show well marked pillow structures. Wherever pressure or movements have affected them, as is the case near the contact with the granites west of the lake or near the vein fissures, the greenstones are altered to schists which are often much contorted. All the schists, however, may not be of igneous origin as there are occurrences of very fissile, deep-green schists which by their association seem to be related to later, fragmental forma-

tions, separated by an erosion interval from the greenstones and schists of purely igneous origin. The strike of the schists is usually north and south, with steep dips generally to the west but with many variations in both strike and dip.

Occurring as infolded lenses in the rocks just described and conforming with them in dip and strike are small masses of conglomerate. The extent of the conglomerate areas is very limited, the only one of any considerable size occurring northeast of Amisk lake, where two narrow bands unite; but small lenses, a chain or so wide and ten to twenty chains in length, are found at many points in the district. The conglomerate is decidedly schistose. It contains a large number of pebbles the majority of which are disc shaped fragments of greenstone and schist. Some pebbles of a felsitic rock occur and many quartz pebbles are present. In some lenses, jasper forms the most striking, although never the most abundant, constituent. Enclosed in the conglomerate are a few lenses of the deep green, very fissile schist mentioned above, which seems to be a part of the formation and probably represents a somewhat clayey layer in the original sediment. Apparently, after the consolidation of the lavas of the greenstone formation, there was some erosion and the deposition of a, possibly somewhat local, clastic formation consisting of conglomerates with some fine-grained members. After consolidation, the whole series was closely folded with the production of schistosity in the less resistant bands. Erosion has now removed the upper parts of the folds leaving only the lower parts of the synclines.

North of Amisk lake the belt of basic rocks is cut off by a widely developed granite gneiss. This is intrusive into the schist and greenstone and may also intrude the conglomerate, although no contacts of the two rocks have been found. The very fresh appearance of the gneiss and the absence of pebbles of gneiss in the conglomerate seem to indicate rather strongly that the gneiss is younger than the conglomerate series. In appearance the gneiss is a strongly banded, light red to light gray rock, with small deep red garnets abundantly developed in places.

A light red, very fresh, massive granite occurs as small masses and dykes in the Amisk lake area and in greater development near Athapapuskow lake and in the region to the east.

The lack of foliation in this rock indicates that it may be of later age than the granite gneiss described; but further examination is necessary to be certain that the gneissoid structure is not an original one, for, if this should prove to be the case the gneissoid and massive granites may be representatives of the same parent magma.

The four rock types described in the preceding paragraphs are, from an economic standpoint, the important members of the Pre-Cambrian complex, although others have been differentiated in mapping and some of these other subdivisions, with the extension of the area of prospecting, may prove to be of more than scientific interest. At present however, the known gold-bearing veins are confined to the greenstone and schist. The granitic rocks and the conglomerate are related to the greenstone in such a manner that it seemed wise to describe them briefly.

The country north of the Saskatchewan river has always been an important trapping ground for the Indians and trading posts were early established there by the Hudson's Bay Company. Cumberland House, on Cumberland lake, was established by Hearne in 1774 and he had, previous to that year, spent one winter on Amisk lake. In recent times parties under Messrs. Dowling, Tyrrell and McInnes explored the district for the Geological Survey. Some prospecting and staking was done, probably from ten to fifteen years ago, as claim-stakes at least that old are to be found along the shores of Cranberry lake and other lakes to the east. Apparently little work was done at that time and probably no important discoveries were made. The present interest in the district dates from the discovery of gold in visible amounts at the Prince Albert claims near the northwest corner of Amisk lake. These were staked in August, 1913, and during the following winter many claims were located around the lake. As a rule the staking has not been carried far from the waterways, though rock exposures are numerous, nor has much real development work been done on the claims staked, with the exception of that at the original discovery of the Prince Albert group. There the vein has been st ipped for some distance, and in October a shaft had been sunk to a depth of seventy feet. A plant is being installed this winter. On some veins discovered in the summer of 1914 near Wekusko lake, on the eastern margin of the belt

of basic rocks several hundred feet of stripping have been done and sinking is to be undertaken this winter. With the exception of the sinking and stripping just referred to no real development work has been done on any of the properties.

The main vein of the Prince Albert claims may be taken as the type of the gold bearing veins of the section. It lies in a zone of schist two hundred feet wide, in massive greenstone. The strike of the schists is north and south and the dip is 60° to the west. The vein conforms in attitude with the enclosing schists. It is lenticular in form varying from nine feet to two feet in width in various parts of the outcrop exposed by the strippings. A parallel vein twenty inches in width lies a few feet to the east of the main vein and, having a somewhat flatter dip, joins the main vein in depth. The gangue is a somewhat sugary, white, pink, blue or mottled brown, quartz. The quartz shows greenish lines and many fracture planes, some of which are coated with a yellowish micaceous mineral. In many cases visible gold is found in flakes along these dark lines or on the fracture planes but sometimes it occurs in apparently unfractured quartz As a rule the specks of gold are fine. Values are also said to be carried by the rusty, altered schists near the vein. Owing, perhaps, largely to the parallelism of the vein to the foliation of the schists, the country rock has not been altered to any great distance from the main fissure. Many narrow stringers run between layers of schist in a direction parallel to the main vein, finally either uniting with it or dying out in a short distance. Calcite is present in minor amounts as a gangue mineral in the vein. Metallic minerals are not present in any great amount in the Prince Albert vein but in other veins occur plentifully. In the Prince Albert deposit, arsenopyrite is the most abundant metallic mineral, occurring chiefly in the altered schist of the country rock as small, well-formed crystals. This mineral is found also in the quartz and is then massive and granular in character. Some pyrite and a little chalcopyrite are present. Galena and stibnite are present but are rare. Molybdenite is fairly common in masses consisting of tiny scales that give it a finely granular appearance. In the upper parts of the vein, where surface waters have had an opportunity to attack the original minerals, there is a slight stain of copper carbonate from altered chalcopyrite. In some veins tiny

fractures were observed filled by a thin film of a mineral, probably a hydrated oxide of iron, that presents almost the colour and lustre of native copper.

No positive statement can be made regarding the origin of the veins. They are certainly Pre-Cambrian in age, since the Ordovician limestones show practically no fracturing. The period of ore deposition is probably late Pre-Cambrian occurring later than the intense folding that produced the foliation in the schists and the schistosity in the conglomerate, as the veins do not show the intense shattering and squeezing that such profound regional metamorphism would cause. By analogy with similar deposits in other places it seems reasonable to assume that the veins are closely related to the later instrusions of granite which are in the neighbourhood of the deposits and which seem to be somewhat similar to granites known to have genetic relationship to gold deposits of other camps. Until further study has been made of the character of the vein filling, and the granite, it is possible to make the statement only, that there is in the area a granite, often showing pegmatitic facies, which seems capable of having produced gold-bearing veins as a dying manifestation of the igneous activity. If more detailed examination shows that this theory is true, it will be a valuable guide to the proper prospecting of the district.

Most of the other veins observed are very similar in general character to the one just described. A small vein on the eastern side of the big point in Wekusko lake occurs near the contact of green schist and a very coarse-grained, dark-grayish porphyritic rock which carries phenocrysts of feldspar some of which are an inch in length. This rock is intrusive into the schist. A short distance farther east the country rock is a garnetiferous and staurolitic gneiss with the same strike and dip as the schists to the west of it, but evidently representing an old sedimentary series. So far as known, no auriferous quartz has been found in this rock. In this section of the region, large pegmatite dykes are rather common and are probably genetically related to the granitic intrusions.

With reference to the possibilities of the country as a gold producing district, it is sufficient to emphasize the necessity of careful development of promising veins both in depth and along

the strike in order that an accurate knowledge of average values and amount of ore may be obtained. Too often plants hurriedly installed before the nature and extent of the deposit are determined, later prove quite unsuited to the requirements. There is especial danger of this in cases where the veins are lenticular in character and the values probably not uniformly distributed throughout the whole mass of the vein. On the whole, the country has not received any thorough examination even of the rock outcrops; and until the holders of claims show sufficient faith in their locations to develop them instead of waiting for their more energetic neighbours to prove the presence of payable deposits, the possibilities of the country will remain largely unknown.

At present prospecting is directed chiefly to the search for gold, but the completion of the Hudson Bay Railway may make the development of other mineral industries possible. Among the possible resources of the district may be mentioned, abrasives, mica, feldspar and structural materials including brick, lime and building stone. Although in the localities seen the garnet content of the garnetiferous gneisses is not sufficiently high to make them a commercial source of garnets for abrasive, it is possible that at some place along the contacts of the granites and older rocks such a deposit may be found. Some years ago a few hundred pounds of mica were taken from pits near Wekusko lake and carried by canoe to Pas. These pits were not seen and nothing is known to the writer, as to the grade or quantity of mica in them. Mica is of course common in these, as it is in most Pre-Cambrian rocks, but it was nowhere seen in commercial quantities. It has been mentioned that in the eastern part of the belt, pegmatitic dykes occur. Some of the dykes are of large size and examination may show that they contain feldspar suitable for quarrying. The thick mantle of lacustrine clays over the eastern part provides an inexhaustible supply of clay quite suitable for the manufacture of brick. Records of tests made on samples of this clay are given in Memoir No. 30, p. 137.* The whole area is well supplied with building stone of excellent quality that requires only a market to make it valuable. The limestone of the escarpment at the southern border of the Pre-Cambrian

*Memoir No. 30, G.S.C. W. McInnes.

occurs in beds of almost any required thickness. There are many places where quarrying would be an easy matter and in most places the amount of overburden to be removed is very small. Also the limestone, although rather highly magnesian, is suitable for the manufacture of lime. The granitic rocks, especially the massive, later granite, would make excellent building stone. It has an attractive light red colour, is of even texture and is capable of taking a high polish.

To sum up it may be said that the discovery of gold-bearing veins in this area of basic Pre-Cambrian rocks makes it an attractive region for further prospecting. At the same time non-metallic minerals may prove to be also of great importance and the search for gold should not cause these other very useful, though more prosaic deposits, to be neglected.

STRUCTURAL GEOLOGY OF THE ALBERTA OILFIELDS *

Problems in the Stratigraphy

By D. B. Dowling, Ottawa, Ont.

Annual Meeting, Toronto, 1915.

The interest that has been aroused in prospecting for oil, in the foothills of Southern Alberta and in the oil possibilities of the known gasfields situated in the less disturbed areas, called for a much closer examination of the structure, thickness and composition of the underlying rocks of the region than had hitherto been made. The areal geology of the larger part of the great plains was outlined by Dawson, McConnell and Tyrrell, between the years 1881 and 1885. The foothill area was not critically examined at that period owing to the time which would have been required for its proper study, and the difficulty that was found in recognizing in the foothills the divisions that had been adopted in the mapping of the formations of the plains. This was due in great measure to the paucity of the exposures in continuous sections of beds of the lower divisions of the Upper Cretaceous. Since the pioneer work on the plains was published, the beds which form continuations south into Montana have been critically examined. The Canadian beds arch over the end of a flat anticline which rises to the south; and two sections in Montana, one on each side of the anticline help to explain some of the difficulties previously encountered in the mapping of the measures found in the foothills.

The principal difficulty experienced in correlation was in connection with the measures known as the Montana group. The beds of this group did not seem to agree; in the two Montana sections in a measure they showed the same differences that are encountered in comparing the foothill measures with those of the plains.

* By permission of the Director of the Geological Survey.

The Montana group in Dakota and Manitoba consists of a series of shales of marine origin. In the eastern Montana section, two sandstone members appear, leaving a shale member at the top and another near the bottom. In the section in western Montana the lower shale is not found and the upper shale is reduced in thickness. The eastern and western sections have been correlated by tracing them to the International Boundary and making connection by means of the formations on the Canadian side.

The nomenclature adopted for the divisions of the Upper Cretaceous in various parts of western Canada and in the United States is given in the table on page 166.

These formations are found in the foothills and on the plains at other localities, and from the characters of the deposits shown and the several sections, it is evident that in any section that will be examined in the Montana foothills west of Cutbank, we may expect the probable absence of the upper shale member and the appearance of additional sandstone members in the shales of the Benton beneath. Sections to the east will probably show a gradual diminution of the sandstone members. As the area under consideration lies to the north, similar variations in the deposits, no doubt, occur and a few notes on the sections there observed are here introduced.

Dakota.—The Dakota sands which contain gas at various points, heavy asphalt oils on the Athabaska and some light paraffin oils in the vicinity of Calgary, are probably represented in the Sweet Grass Hills by one of several sandstone beds which apparently lie at the base of the Benton. To the west these sandstone beds thicken as the mountains are approached and in Blairmore section on Crowsnest river the basal beds lying above the coal measures of the Kootenay are upward of 2,000 feet in thickness. To the north, they are less than 1,000 feet in thickness, in the foothills west of Calgary, and decrease to 200 feet on the Athabaska at some distance from the mountains.

Colorado Group.—The Colorado group, mainly shales with an occasional sandy member, underlies probably the whole of the plains and is found inside the mountains. Its thickness in

TABLE OF FORMATIONS

	Western Montana	Southern Alberta			Central Montana	Manitoba	South Dakota
			West Forks of Milk R.	East Near Pakowki L.			
Montana Group	Bearpaw Shale (*Marine*)		Bearpaw Shale (*Marine*)	Bearpaw Shale (*Marine*)	Bearpaw Shale (*Marine*)	Odanah Shale (*Marine*)	Foxhill Sandstone (*Marine*)
	Two Medicine formation (*Fresh and brackish*)	Belly River formation	"Pale" and "Yellow" beds (*Fresh and brackish*)	"Pale" and "Yellow" beds (*Fresh and brackish*)	Judith River formation (*Mainly fresh water*)		
			Shales at Forks of Milk River	Shales in Coulee near Pakowki (*Marine*)	Claggett Shale (*Marine*)	Millwood Shales (*Marine*)	Pierre Shales (*Marine*)
	Virgelle Sandstone		"Castellated" rocks	"Castellated" rocks of Milk River	Eagle Sandstone		
Colorado Group	Benton Shales (*Marine*)	Lower Dark Shales (*Marine*)			Benton Shales (*Marine*)	Niobrara Shales (*Marine*) Benton Shales (*Marine*)	Niobrara Shales (*Marine*) Benton Shales (*Marine*)
Dakota Group		Exposures of sandstone in the Sweet Grass Hills, and also reached in borings in Alberta.			Exposures in Sweet Grass Hills		

the Sweet Grass Hills is estimated to be 850 feet, and in the vicinity of the mountains at Blairmore, 2,700 feet. In the foothills the thickness may well be over 2,000 feet.

Montana Group.—The character of the deposits of the Montana group, as shown in the section above, indicates that the sea margin changed several times but remained during this period within the area now occupied by the foothills and the plains, and that in tracing these beds eastward a diminution in the thickness of the sandstones and an increase in the shales may be expected. Northward the problem of the variation in the deposits is being studied and a few points that have been determined may be referred to.

It is fairly well established in the section between Medicine Hat and the mountains that the sandy beds at the base are very thin at Medicine Hat, and in the foothills resemble the castellated rocks of the southern section. Northward they evidently become thin again, as they do not appear in the Athabaska section, and may be represented in the mountains by the Brazeau formation or the Dunvegan sandstones of the Peace river. The marine shales, called 'Claggett' in the eastern Montana section, lying between the sandstone members, are probably represented at Medicine Hat by a much thicker shale series. In the foothills near the Crowsnest river this shale series has thinned out and can not be recognized. It is probably present in the sections north of the Highwood river, increasing in thickness to the north-east, while the sandy beds of the underlying lower part of the Belly river series decreases materially and ultimately thins out. This shale would then in the absence of the underlying sandstone, represent the series overlying the Colorado shales of the Athabaska river and of the northern foothill exposures. The upper sandstone and clay member of the Belly river formation is thicker in the west than in the east, and the section west of the Porcupine Hills resembles that in western Montana in the inclusion in it of all the Belly river rocks in one thick sandstone member.

The Bearpaw shale, the upper part of the Pierre, is a marine deposit that overlies the Belly river beds. It is exposed over large areas in southern Alberta and in a somewhat diminished form is found in the foothills to the south. Its thickness is apparently maintained westward but the marine portion is confined to the

central part of the section. The beds at the base, in the vicinity of Lethbridge and even farther east, contain coal-bearing zones, which in places have valuable coal deposits. In the foothills coal seams are found both at the top and the bottom of these shales and it is suspected that north of Highwood river the marine deposits of the formation are very thin, if they are represented at all, so that the division between the Edmonton sandstones and the 'pale beds' of the Belly River formation may be marked farther north, by merely a coal-bearing zone or transition beds, instead of by marine shales. Some indication of this is given in the character of the rocks in the section on the Saskatchewan river, east of Edmonton. The sandstone and clays of the Edmonton formation do not end abruptly, but merge by degrees downward into a shale series. The shales overlie sandstones which may possibly represent the top of the Belly River series, there composed of a thin sandstone member. The shales beneath are no doubt directly continuous with the Pierre of the east.

The geologist working in the foothills of central Alberta must therefore be prepared to find, not the well-marked formations found in other places, but a great series of sandstones, ranging in age from the earlier deposits of the Montana group, up to Tertiary deposits. There may be but traces of the marine beds which are found in the sections of the plains to the east. It may be noted also that the Benton formation is well developed and forms the great shale series of the foothills, but that north of a line drawn north-west from the forks of the Highwood river, shales, belonging to the lower part of the Montana group, cap the Benton.

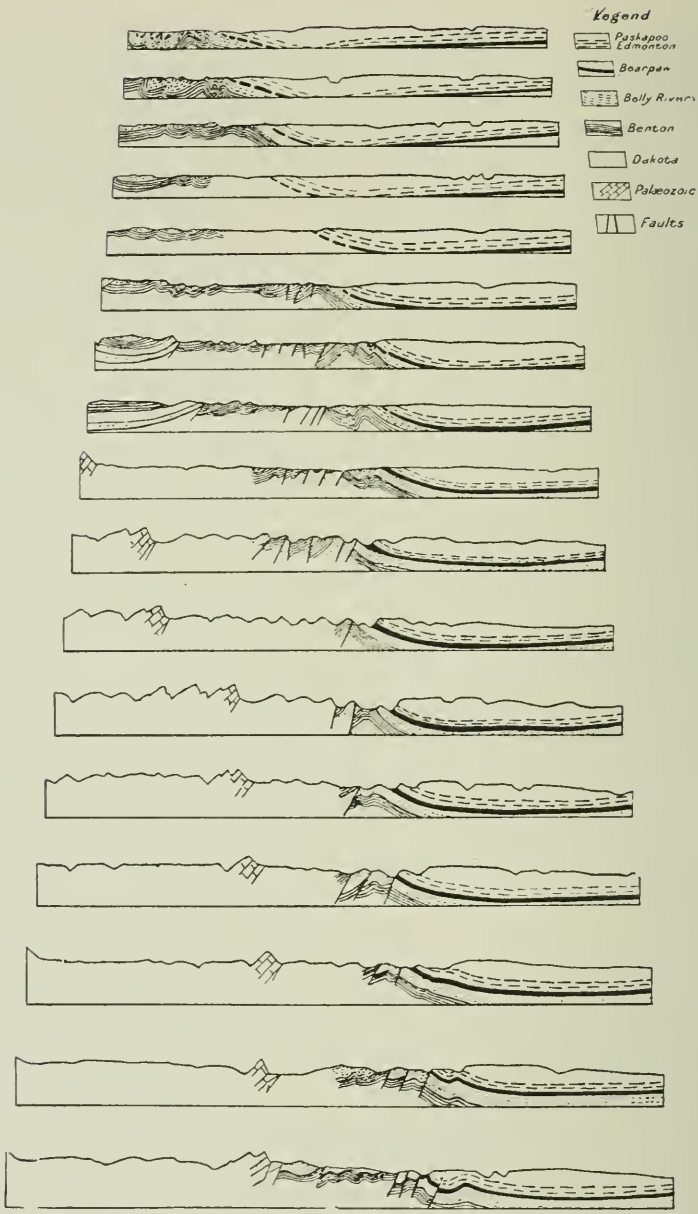
STRUCTURE

The structure of the outer portion of the foothills has been partially mapped and a comprehensive view of its general character may be gleaned from the accompaying sketch and sections. It will be seen from them that there was an uplift of the lower measures toward the mountains accompanied by profound fracturing throughout the disturbed zone. As the lines of fracture penetrate below the beds containing the possible oil supply, the fault blocks have necessarily a limited oil drainage area and do not afford promising ground for wells. Our field

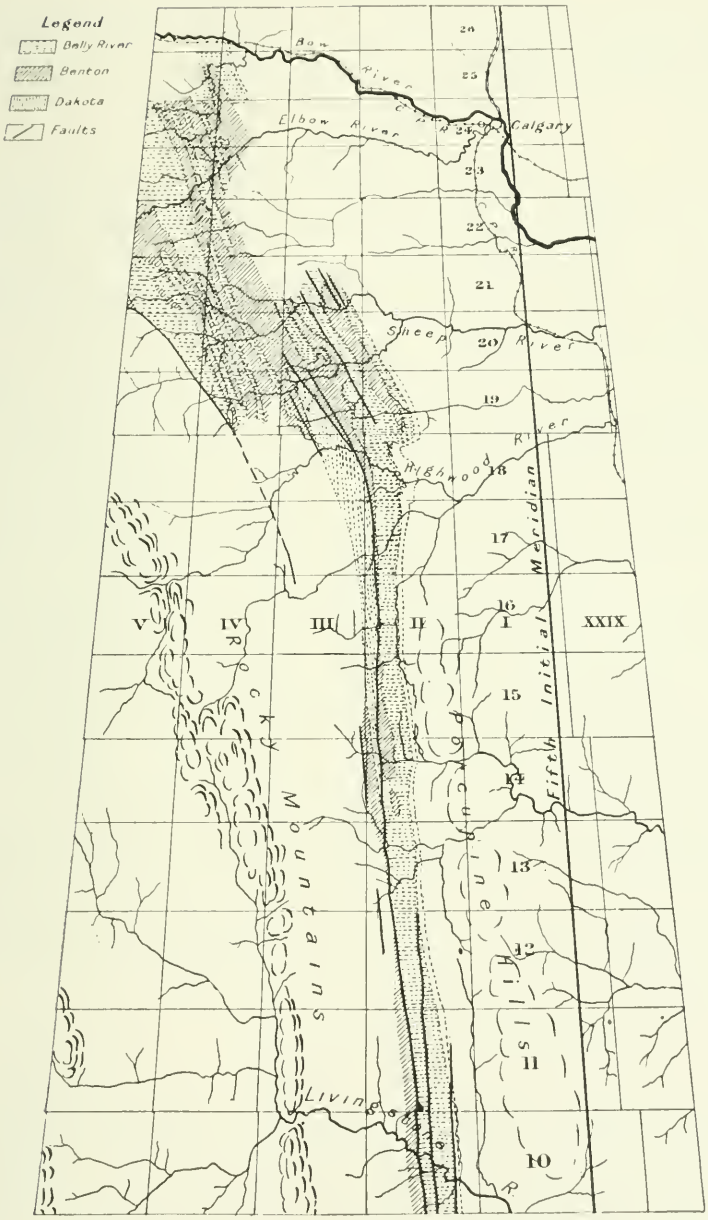
of study has been limited therefore to the eastern edge of the broken country in the hope of finding anticlines in close connection with the less disturbed beds of the Alberta syncline which lies to the east. One of these on Sheep Creek is being thoroughly prospected and one well is now producing a small quantity of very light oil. Another in Tp. 23, west of Elbow river, has yielded heavier oil.

In the country to the east of the broken area and syncline indicated in the accompying sketch and sections, the beds are so slightly flexed that they seem at any one point to be almost horizontal. They are as a rule less consolidated than the beds near the mountains and the rivers are deeply trenched. This trenching of the rivers is in nearly every case accompanied by a series of land slips extending back for some distance from either bank. This already has been interpreted as faulting by several 'experts,' and an intricate structure, showing anticlines and faults has been pictured, providing many 'Oil' companies with attractive prospectuses. There is a wide anticline, however, in Southern Alberta, between the outer foothills and the Cypress Hills, at the eastern boundary of the province which extends from northern Montana well into Alberta. This had already been the subject of investigation for a possible natural gas reservoir and the wells at Bow Island which supply Lethbridge and Calgary are located on it. Attention has again been called to it by the discovery of slight signs of oil in springs on the slopes of the Sweet Grass Hills in Montana, and several drilling rigs have been placed in the valley of Milk river and even at the Boundary line on the flanks of the above hills. The borings in this vicinity will probably penetrate the sandstones of the lower part of the Belly river series and also the Benton before reaching the Dakota from which there seems to be some chance that gas at least will be obtained. The thickness of the Cretaceous measures is here less than in the foothills, and very deep wells will not be necessary to test the ground.

A flat anticlinal structure is also indicated by the outcrop of the Belly river rocks in the eastern part of Alberta. This anticline runs in a north-west direction and is crossed by several stream valleys, notably that of the Battle river. The Grand Trunk Pacific railway crosses the Battle river near the axis of the anticline. A well has been sunk for gas near the railway, to the



Structure Sections at Township Lines. See Diagram.



Perspective Diagram of Part of the Foothills.

west of the centre of the anticline and struck a small gas reservoir at a depth of 2,340 feet.

DEVELOPMENT

Oil seepages have been known for many years in the mountains along the International Boundary, east of the Flathead valley. Several companies bored wells at the outer edge of the mountains, and about six years ago there was some excitement over the discovery of oil in a well near the Waterton lakes. The difficulty of getting machinery to this region and the probability of the area being limited, prevented extensive prospecting. The finding of oil last year in less broken country at the outer edge of the foothills in an easily accessible area at once attracted the attention of the speculative element of the population and many companies were formed and oil leases applied for. The discovery well is situated on an anticline of Benton shales flanked on both sides by sandstone ridges cut through by the valleys of three streams. As the sandstone at the crown of the anticline has been removed by denudation the direction of the anticlinal axis is marked by a series of transverse valleys eroded in the shales. These depressions afford favourable locations for derricks and eleven wells are now being bored. In the country to the west of this anticline many other drillings are being made so that in the portion of Alberta shown on the diagram of sections there were during 1914 about 36 separate points of attack mainly in the foothill belt. Two wells have reached depths of more than 3,000 feet without success. Seven, including the discovery well, are over 2,000 feet deep. Fourteen are over 1,000 feet deep and thirteen have reached depths less than that footage.

Three companies are boring in the Milk river valley, and four in the foothills north of Bow river.

In a few cases it may be considered that the ground has been found to be barren of productive reservoirs; but in the majority of cases the mechanical difficulties have been so great, owing to the depth to the prospective oil sands, that no positive result has been reached. In some cases too, the wells have been badly located from the view point of structure.

In the discovery well, light gasoline oil and a heavy gas flow was found at 1,550 feet, in sandy beds in the lower Benton. At 2,700 feet another flow of gas and oil was found in the Dakota or in sands of about that or slightly lower horizon. This oil was also light in specific gravity (about 55° Baumè), and was accompanied by a heavy flow of gas which has been shown by experiment to produce a light gasoline on condensation.

It is claimed that showings of oil have been got in several wells in the vicinity.

A discovery of oil, 40° Baumè, in the well of the Moose Mountain Oil Company was announced on November 24th, 1914. The well has since been shot and a yield of 25 barrels per day is claimed. The oil is dark brown and shows a greenish colour by reflected light. It was struck at a depth of 1,690 feet in the top beds of the Dakota.

NOTES ON THE GEOLOGY AND EXPLORATION OF COPPER MOUNTAIN IN THE SIMILKAMEEN DISTRICT OF BRITISH COLUMBIA.

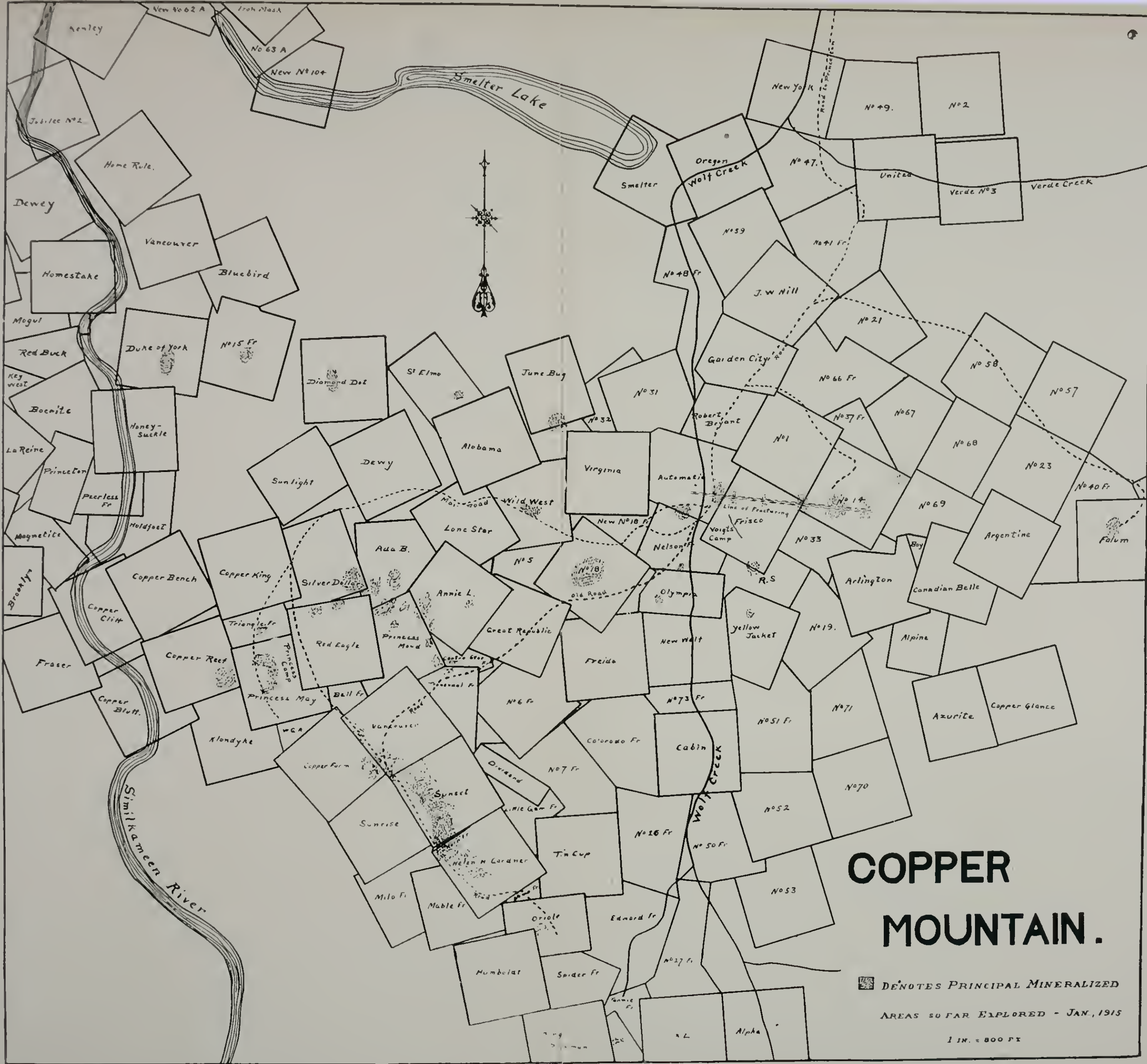
By Frederic Keffer, Spokane, Wash.

Annual Meeting. Toronto, 1915

Copper Mountain, in the Similkameen Division of Yale District, British Columbia, lies to the south of the town of Princeton, the centre of that section of the camp now under active exploration being about 13 miles from the town. The elevation ranges between 3,000 and 4,500 feet, the majority of the mining claims being between 3,500 and 4,200 feet.

The deep valley of Wolf Creek traverses the centre of the camp from south to north, whilst on the west the Similkameen River, which here runs through a deep and narrow gorge, divides Copper Mountain from Kennedy Mountain. Except in the river gorge, the country is easily traversed, the slopes as a rule being gentle. A forest of pine and fir covers most of the camp, and prospecting is rendered difficult on account of the covering of soil and glacial drift which ranges from a few feet to twenty or more in thickness. Water at present is scarce, but a supply will ultimately be obtained by pumping either from the river or from Wolf Creek. With its moderate elevation, climatic conditions at the camp favour mining work throughout the year.

The country rocks of the greater portion of the camp have been classed by Charles Camsell as belonging to the post-Paleozoic age, and consist principally of granodiorites and monzonites. These rocks, however, include the remnants of the older paleozoic limestones, argillites, etc., which paleozoic rocks, for the most part, have been so completely assimilated by the later batholithic flows of granodiorite and monzonite, as to have lost almost entirely their original character, there now existing but few and small remnants of unaltered paleozoic rocks. The batholithic mass itself has undergone extensive alteration, it rarely happening that one finds an entirely fresh specimen. The predominant minerals are feldspar, both orthoclase and plagioclase, the latter predominating, and augite, with hornblende probably derived therefrom; biotite, hematite, calcite.



COPPER MOUNTAIN.

DENOTES PRINCIPAL MINERALIZED AREAS SO FAR EXPLORED - JAN., 1915

1 IN. = 800 FT

magnetite and the sulphides of iron and copper make up the rest of the batholithic rocks. These minerals enter into the composition of the rocks in greatly varying proportions, lending considerable variety to the different sections of the camp. Toward Smelter lake on the North side of the camp the rocks are tertiary basalts and andesites, in which however little economic minerals have been discovered.

This series of granodiorites, monzonites and altered paleozoics has been cut by a variety of later dykes, the great majority of which are cream coloured quartz-feldspar porphyries, although dykes of syenite, rhyolite and other rocks are occasionally seen. The dykes are infrequent from the river to near the summit of the slope up toward the east. As the summit is neared they form a series which persists to the east margin of the camp. The general direction of the dykes is a little west of north, but locally they vary much indeed and also form a network, which owing to the covering of drift, it has been a difficult task to locate and map. The dip varies from nearly vertical to about 70 degrees toward the east. In the vicinity of the Sunset claim where they exhibit notable differences in texture, they have been classed as dykes A, B and C. The variation seems to have been produced by differing rates of cooling. "A" dyke has large phenocrysts of quartz in a fine grained ground mass of feldspar. "B" dyke has much fewer quartz phenocrysts; whilst "C" dyke rarely has any.

In places where a dyke is several hundred feet in width, the centre is generally of "A" type, whilst the margins show "B" and "C" varieties. Plates one to seven herewith illustrate micro-photographs of these quartz-feldspar porphyries, both with ordinary and with polarized light. In the Sunset section of the camp granodiorite predominates between and to the west of the first series of dykes, whilst to the east the prevailing rock is monzonite.

The sulphides of copper appear to be original constituents of the country rocks in many parts of the camp, although taking the body of rocks as a whole these sulphides are far too disseminated to be of any economic value. Deep diamond drilling has quite conclusively proved the above statement to be true. The ores themselves occur under two main conditions: First,

they are found in the neighbourhood of the contacts of the batholith with the original paleozoic rocks where fracturing both of the batholithic mass and of the partly or wholly digested paleozoics has afforded ingress to copper-bearing solutions, derived from the deeper lying batholith. Secondly, where these paleozoic rocks are little in evidence in other parts of the camp, mineralization has followed lines of fracturing between and usually at right angles to the porphyries. These porphyries themselves, in no case yet observed, have been mineralized with copper sulphides; neither are they fractured, even along the fracture planes of the rocks lying between. This circumstance has made it somewhat doubtful whether the intrusion of the dykes preceded or followed ore deposition. The fact that the best mineralized ground is usually along or between dykes points to the shattering of the batholith by the intrusion of the porphyries prior to mineralization, and therefore, perhaps the weight of the argument is in favour of the latter assumption. There has been but slight reaction between the porphyries and the batholith so that the margins of the dykes are usually very clearly defined.

Certain broad distinctions as to mineralization can be traced in the camp. In the southwestern part of the camp the copper minerals at or near the contacts of batholith and paleozoics are usually chalcopyrite and bornite, microscopic examination showing the latter mineral to be primary and probably formed coincidently with the chalcopyrite. These conditions obtain in the Sunset and surrounding mineral claims. Further north there is much pink feldspar with the granodiorites and monzonites, and the copper occurs solely as chalcopyrite. Magnetite is also much more abundant, as is instanced in the Ada B., Silver Dollar and adjoining claims. And also in the Duke of York, No. 15 Fraction, Diamond Dot and June Bug claims.

Toward the northeast and across Wolf Creek in Voigt's Camp, where mineralization in one section follows a strong line of fracturing extending through the Automatic, Frisco and No. 14 claims, there occurs much hematite, calcite and pink feldspar, as gangue minerals in the ore. The copper is chalcopyrite with only occasional admixture of bornite, and the gold contents of

the ore is markedly higher than elsewhere on Copper Mountain. The gold follows the hematite rather than the chalcopyrite. Masses of pure hematite run as much as $1\frac{1}{4}$ ounces in gold whilst masses of almost pure chalcopyrite contained but traces of the precious metal. In the No. 14 claim, the ore consists of a mass of little angular fragments of rock cemented by calcite, and presents quite a contrast to the other ores of the camp.

As a rule, the copper minerals of the camp are quite uniformly distributed through the ores in little veinlets occupying the fissures; but occasionally, as in the Triangle Fraction, the ore occupies a fissure of from one to three feet in width, and consists largely of massive chalcopyrite.

In the Silver Dollar claim, a shaft was sunk 100 feet in a 'chimney' of almost completely kaolinized feldspar, where the copper occurred as chalcopyrite scattered through the rock in quite large crystal masses. The accompanying map of the camp shows the mineral claims and the general distribution of mineralized areas so far explored.

A number of photomicrographs of the rocks of the camp are shown herewith, which illustrate some of their general characteristics.

Plates 1 to 7 have already been noted. Plates 8 and 9 show an intimate mixture of chalcopyrite and bornite from the 175 ft. level of the Sunset workings. Plate 10 shows a 'digested' limestone from the 175 ft. level, now completely altered to feldspar, augite, magnetite and quartz, together with some copper sulphides, these alteration products being themselves much decomposed. Plates 11 and 12 show characteristic sections of the country rock, the specimens being from a diamond drill hole on the Sunset claim. Both are considerably decomposed, the deeper rock being the most affected. No. 13 is typical of the ore found along the Automatic, No. 14 claim, line of fracturing (see map), and consists of innumerable fragments of feldspar now cemented by calcite which also includes much hematite and sulphides of iron and copper.

No. 14 is a similarly crushed mass of now consolidated fragments impregnated with copper and iron sulphides, from the Falum claim.

Plate 15 from the King Solomon claim in the southern part

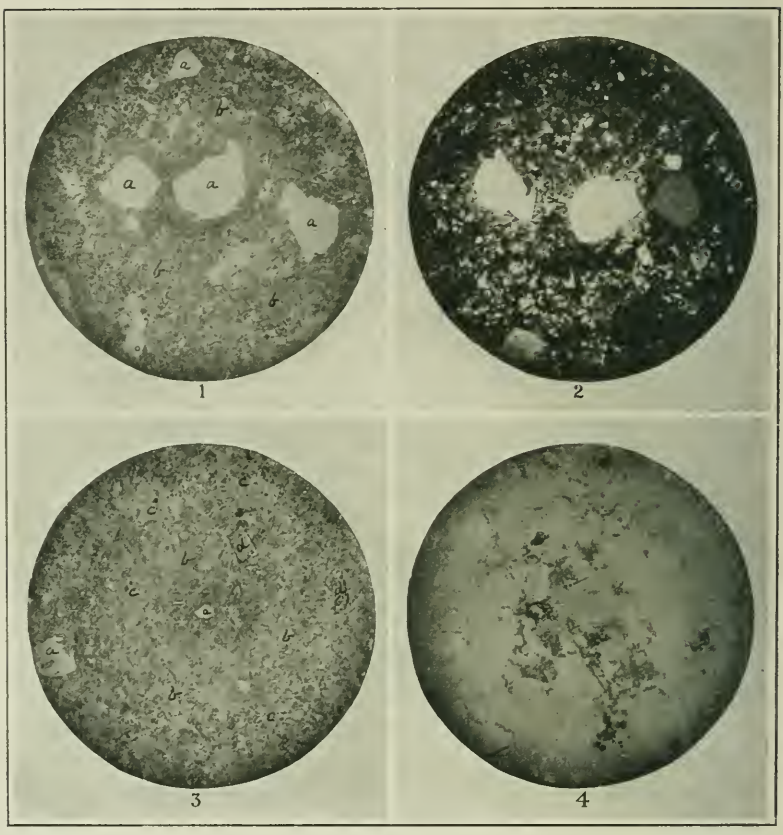


PLATE 1.

PORPHYRY DYKE "A"—
SUNSET CLAIM.

a=Quartz. *b*=Decomposed Feldspar. 20 Diam.

PLATE 2.

DYKE "A."—

Same as Plate 1. Crossed Nicols.

PLATE 3.

PORPHYRY DYKE "B."—
SUNSET CLAIM.

a=Quartz. *b*=Decomposed Feldspar. *c*=Magnetite. *d*=Decomposed Hornblende. 20 Diam.

PLATE 4.

PORPHYRY DYKE "C."—
SUNSET CLAIM.

Shows almost completely Kaolinized Feldspar, with a little Quartz and Magnetite. 20 Diam.

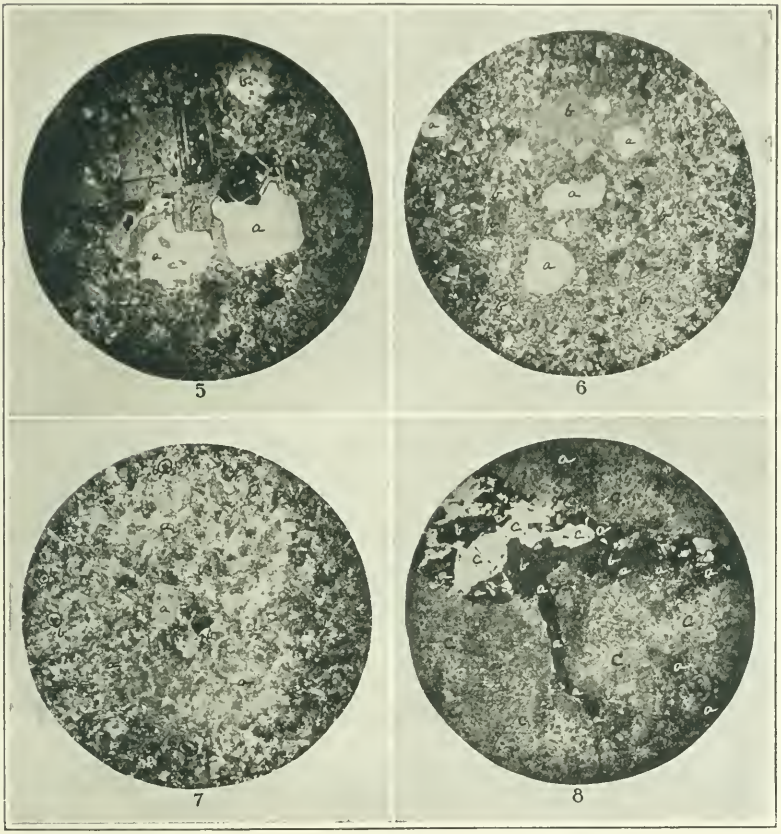


PLATE 5.

PORPHYRY DYKE "A."—NEAR SUNSET SHAFT.

Much decomposed. *a* = Quartz. *b* = Feldspar. *c* = Muscovite. Polarized Light. X Nicols. 20 Diam.

PLATE 6.

PORPHYRY "A" DYKE—ORIOLE CLAIM.

Much Kaolinized. *a* = Quartz. *b* = Feldspar. *c* = Muscovite. Under X Nicols. 20 Diam.

PLATE 7.

PORPHYRY—KING SOLOMON TUNNEL.

Highly Kaolinized. Mostly Feldspars. *a* = Very little Quartz. Many small Crystals of Pyrite. *b* = Some Muscovite. X Nicols. 20 Diam.

PLATE 8.

BORNITE ORE—SUNSET SHAFT 175 FEET.

a = Bornite. *b* = Chalcopyrite. *c* = Much Decomposed Feldspar and Augite. 20 Diam.

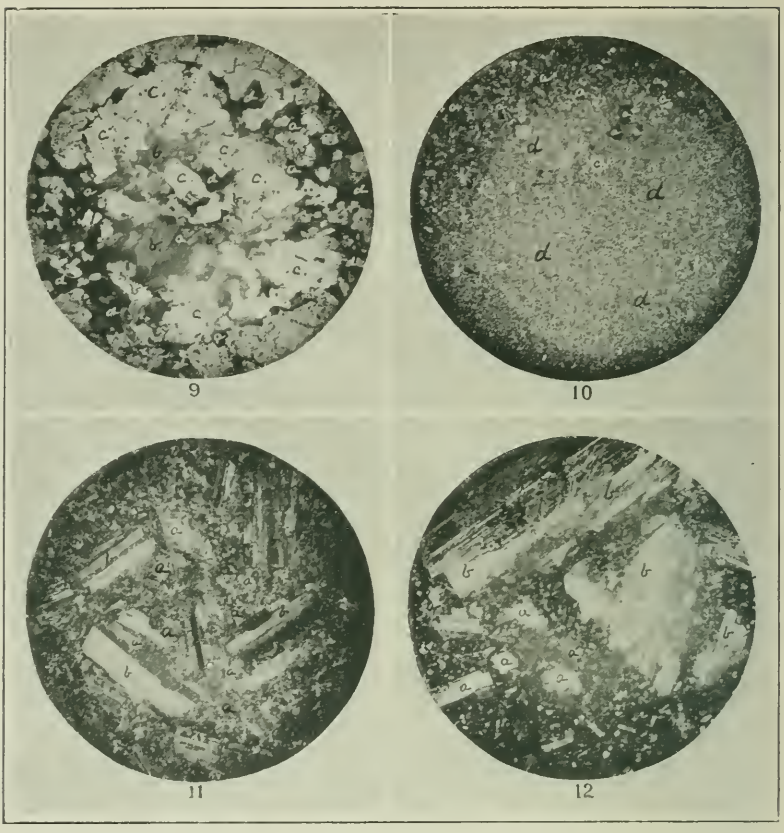


PLATE 9.

ORE FROM SUNSET SHAFT—175 FEET L.

a = Bornite and Chalcopyrite. *b* = Augite. *c* = Feldspar. (*b* and *c* much decomposed). *d* = Chlorite and Biotite, the latter also much decomposed. 20 Diam.

PLATE 10.

FROM S. DRIFT—175 FEET LEVEL—SUNSET.

a = Chalcopyrite. *b* = Magnetite. *c* = Quartz. *d* = Decomposed ground mass of Feldspar and Augite with much Magnetite. X Nicols. 20 Diam.

PLATE 11.

FROM D.D. HOLE NO. 12, AT 5 FEET—SUNSET CLAIM.

a = Ground-mass of entirely decomposed Feldspar and Augite, mainly the former. *b* = Partly decomposed Feldspar. X Nicols. 20 Diam.

PLATE 12.

FROM D.D. HOLE NO. 12, AT 157 FEET—SUNSET CLAIM.

a = Augite, entirely decomposed. *b* = Partly decomposed Feldspar. Many small Crystals of Magnetite present. X Nicols. 20 Diam.

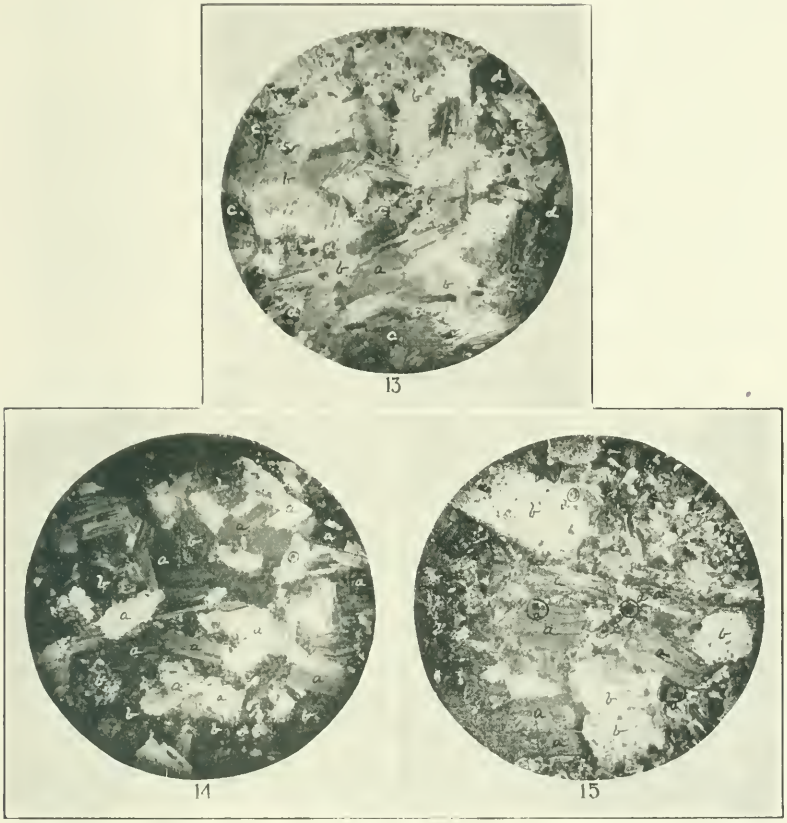


PLATE 13.

ORE FROM NO. 14 CLAIM—VOIGT'S CAMP.

Fragments of Feldspar, *a*; cemented by matrix of Calcite, *b*. *c*=Hematite. *d*=Chalcopyrite and Pyrite. X Nicols. 20 Diam.

PLATE 14.

FROM FALUM SHAFT—VOIGT'S CAMP.

a=Feldspar. *b*=Augite, altered to Hornblende. o=Apatite. Much Magnetite is present. X Nicols. 20 Diam.

PLATE 15.

ROCK EAST OF ORE—KING SOLOMON CLAIM.

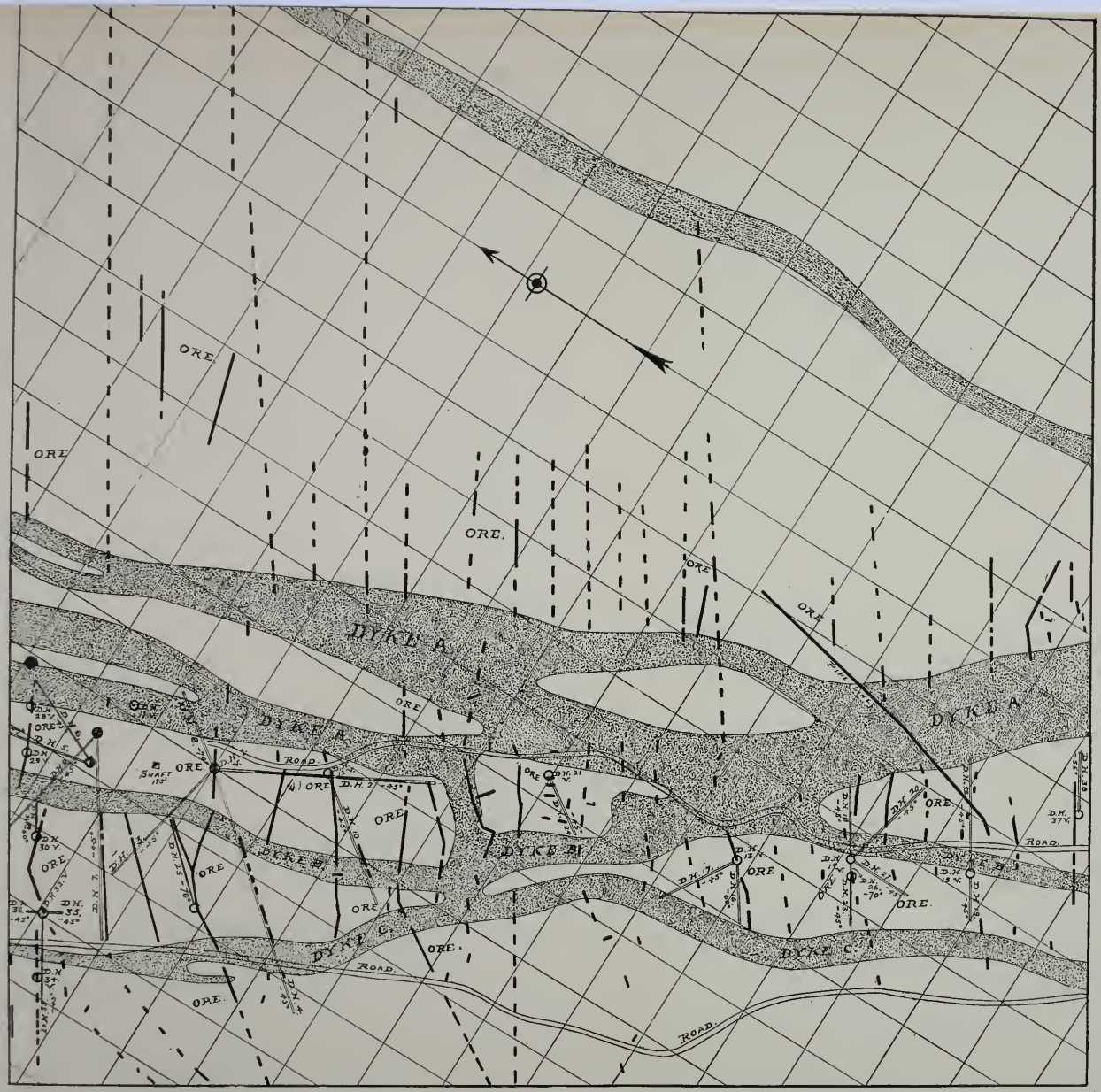
a=Partly altered Feldspar. *b*=Kaolinized Feldspar. *c*=Decomposed Audite. *d*=Magnetite. X Nicols. 20 Diam.

of the camp shows country rock in an advanced stage of alteration, the feldspars being largely kaolinized.

Although the copper impregnation often extends to the bottoms of the deepest holes so far drilled, commercial ores have been limited as to depth, none having so far been found at depths over 500 feet, this probably being due to decreased fracturing with increase of pressure. However, the extensive area of the mineralized ground compensates for limitations in vertical extent, and as exploration proceeds new ore bodies are constantly discovered, operations to date having disclosed an immense tonnage of workable ores.

Exploratory operations have been carried on at intervals during the past fifteen years by Mr. Emil F. Voigt and his associates, who did a great deal of pioneer work in the camp. The greater amount of exploration, however, has been done by The British Columbia Copper Co., Ltd., which Company worked the Sunset and some other claims in 1905-6 and later on from 1911 until the present time has worked both in the Voigt section of camp and in the southwestern portion, known collectively as Princess Camp, comprising the Ada B. group, the Silver Dollar, Princess Maud, Princess May, Annie L., Sunset, Vancouver, H. H. Gardiner, Copper Farm, and a host of other claims, less promising than those named.

Although quite a number of prospect shafts have been sunk and several thousand feet of tunnels and drifts run, still by far the greater amount of the work has consisted of trenching and diamond drilling. Upon bonding a claim, the surface is first thoroughly trenched in order to locate ore bearing areas, dykes and other rock occurrences. Having picked up ore in the course of this trenching, the ore-bearing areas are diamond drilled, the thoroughness of this drilling depending of course on the results attained as drilling progresses. The usual procedure is to first drill a vertical hole to the limit of the ore, and then to follow this with holes drilled from the same set up, in different directions. How many set ups are made and the number of holes drilled are dictated by the circumstances covering the particular mineralized area being explored. Results of drilling in many places have shown that the tonnage of any particular block of ore can be estimated by assuming a block bounded by the surface mineral-



Map of Claim Showing Dyke System and Method of Exploration

1 inch = 1000 feet.

ized area (as disclosed by trenching) and planes connecting the lower limits of the ore as found in the drill holes. Checks on these estimates, however, are made by dividing the ground into parallel vertical sections and estimating the ore between the divers sections. Any plan of estimating tonnages from diamond drill holes in ore bodies of such irregular shape as these, involves considerable uncertainty, and it is customary to keep strictly within the limits of the ore as shown in the holes and surface area in estimating 'practically assured ores' leaving considerable latitude as to probable and possible ores outside the limits of the holes.

Accompanying this paper is a map of one of the principal claims now being explored, showing the great amount of trenching done, the diamond drill holes, and the intricate dyke system. When it is considered that this is but one of many claims similarly explored, the great amount of needful work becomes apparent.

General plans for the exploitation of the camp have been held in abeyance until exploration shall disclose the size and positions of the principal ore occurrences, when a comprehensive plan of development involving the least amount of dead work can be devised. Whether the divers mines will be reached from a centrally located shaft, or from an adit tunnel is not yet decided, but it is probable that an adit tunnel system will be adopted, this tunnel entering the mountain from the Similkameen River slope.

The Copper Company abandoned operations in 1906, partly on account of lack of transportation, but mainly because the ores proved too silicious to be smelted directly, and ordinary water concentration was found to be impracticable owing to the low recovery attainable. Since that date, however, the flotation process has come into extensive use, and the results of many tests have shown that the Copper Mountain ores are quite amenable to the process with but little loss in the tailing.

The high excess of silica over iron of from 40 to 60 per cent in the ores, is reduced to about 10 per cent, whilst the sulphur contents, always low in the ores, is correspondingly increased. Lime, which in the ores is lacking, it seldom running over four per cent, is almost entirely eliminated in flotation, and the concentrates will therefore need to be smelted in conjunction with lime bearing ores, such as are found in the Boundary District.

AINSWORTH MINING CAMP

STUART J. SCHOFIELD, Ottawa, Ont.

Annual Meeting, Toronto, 1915

The Ainsworth mining camp (Plate I), formerly called the Hot Springs camp, is situated on the west side of Kootenay lake, B.C., about 2 miles north of its outlet into Kootenay river and 22 miles from Nelson, the capital of West Kootenay district. Its position relative to the other mining camps is shown in the accompanying index map (Figure I). The camp may be said to include the territory between Coffee and Woodberry creeks—(a distance of 4 miles in a north and south direction); and from the west shore of Kootenay lake to the eastern edge of the West Kootenay granite batholith (Nelson granite)—a distance of 3 miles in an east and west direction.

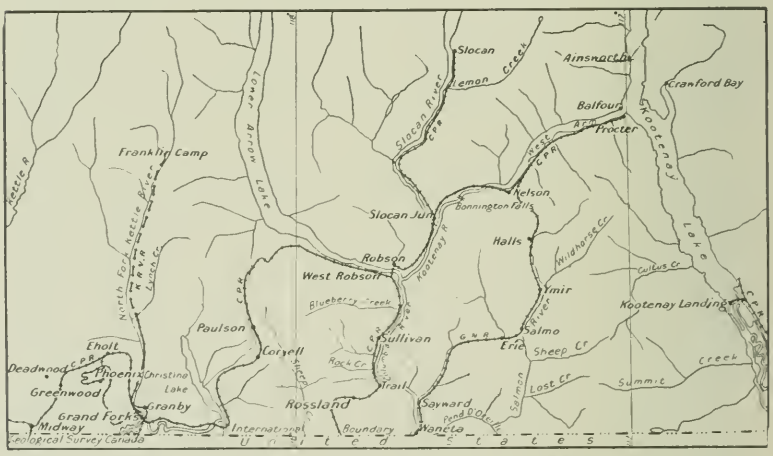


Fig. I.—Index Map showing the position of Ainsworth.

Although the Blue Bell mine which is situated across the lake from Ainsworth is reported to have been discovered by a

Scotch botanist named Douglas in 1825, no locations were made at Ainsworth until 1883, when Thomas Hemmill staked the Lulu and the "Springs." During the same year the Surprise, Morning Star and Evening Star were located along what is now known as the Spokane vein[1]. Since that date numerous other claims have been staked. Development has proceeded continuously since 1883, but it was not until last year that systematic exploration of the ore bodies was undertaken when the Consolidated Mining and Smelting Co. of Canada acquired several properties.

[1]Plate I.—Ainsworth from above the Blue Bell Mine, looking west. Riondel in foreground.

The development of the district is shown by the following table of production. The production of the Blue Bell mine, is included in the list.

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[1]G. M. Dawson, Geol. Surv. Can., Ann. Rep. 1888-1889.

PRODUCTION[1] OF AINSWORTH MINING DIVISION

Year	Tons	Gold, Lode	Silver		Lead		Zinc		Totals for
		Ounces	Ounces	Value	Pounds	Value	Pounds	Value	Division
				$		$		$	$
1896	.....	...	374,097	250,665	3,186,592	94,961	.....	.....	345,626
1897	5,556	...	524,578	313,697	3,543,237	126,848	.....	.....	440,545
1898	17,380	...	167,147	92,515	1,978,297	67,262	.....	.....	159,801
1899	3,760	91	268,165	151,781	3,588,577	144,261	.....	.....	297,930
1900	5,313	28	352,167	205,454	3,366,962	143,433	.....	.....	349,465
1901	5,938	63	324,913	181,951	3,788,412	147,748	.....	.....	331,011
1902	4,939	5	320,719	158,916	3,083,039	112,839	.....	.....	272,967
1903	24,332	33	108,678	55,187	4,299,727	163,949	.....	.....	219,818
1904	14,569	2	90,004	48,026	3,091,648	119,956	.....	.....	168,023
1905	3,331	28	99,781	57,204	1,002,114	42,490	.....	.....	100,273
1906	19,431	19	165,915	105,273	3,173,353	161,524	.....	.....	267,190
1907	17,781	118	301,322	187,000	3,654,775	175,429	.....	.....	364,868
1908	38,282	162	314,142	157,762	4,790,216	181,070	.....	.....	342,181
1909	97,698	162	352,555	172,505	10,298,343	396,486	.....	250,000	867,340
1910	21,850	71	233,010	118,397	2,558,353	102,334	2,083,896	95,859	318,058
1911	671	4	77,375	39,183	289,009	11,502	.....	.....	50,768
1912	32,741	80	301,755	174,384	4,863,894	195,723	.....	.....	371,760
1913	92,472	25	447,015	253,905	9,027,861	354,795	150,680	7,233	616,450

[1]Reports of Minister of Mines for British Columbia.

Production in tons of Ainsworth, reported at Trail by C.M. & S. Co.

July	1903–1907 (inc.)	1908	1909	1910	1911	1912	1913	1914
Highlander	144	.....	.....	.....	.....	.....	.....	.....
Gallagher	79	19	73	.....	.....	.....	10	.....
Krao	995	.....	.....	.....	.....	.....	.....	.....
Maestro	201	126	65	35	113	.....	157	1,232
New Jerusalem	19	31	.....	.....	.....	.....	.....	.....
Highland	4,334	.....	198	304	22	.....	1,130	2,915
No. 1	324	19	21	.....	137	524	4,017	4,885
Spokane	493	13	.....	.....	.....	.....	.....	.....
United	182	.....	.....	.....	.....	.....	.....	.....
Pontiac	.....	.....	30	.....	.....	.....	.....	.....
Jessie Bluebird	.....	25	.....	35	.....	.....	.....	.....
Ayesha	.....	.....	.....	.....	10	.....	.....	.....
Silver Hoard	.....	.....	.....	.....	.....	202	1,287	43
Florence	.....	.....	.....	.....	.....	62	16	.....
Bluebell	1,663	2,229	4,907	1,262	.....	3,720	7,209	4,754
Early Bird	.....	.....	.....	.....	.....	.....	.....	57

ORES

The ores mined at Ainsworth consist essentially of two varieties.

(1) Oxidized earthy ore containing native silver as in the No. 1 mine. The values of the No. 1 ore for the years 1912, 1913 and 1914 as given by the Annual Report of the Consolidated Mining & Smelting Co. are as follows.

	Oz. Ag per ton	% Pb
1912	95.7	3.22%
1913	37.8	1.68
1914	32.0	1.045

(2) Primary ores consisting of galena, zinc blende, and iron pyrites, as seen in the Highland mine. The ore is a milling proposition. The values of the concentrates are as follows:

	Oz. Ag per ton	% Pb
1913	15.4	55.65%
1914	25.6	78.17%

The ratio of concentration [1] varies from 1 to 7 to 1 to 10 depending on how carefully the work is performed.

[1] Report, Minister of Mines, B.C., 1914, p. 149.

General Geology

The Selkirk Mountain range which the Ainsworth Mining district is situated, forms the western part of that great folded mountain system which was built at the close of the Jurassic or in early Cretaceous times.[1] The eastern portion of this system, the Purcell range[2] is separated from the western portion by the Purcell trench which is in great part occupied by Kootenay lake. The Rocky Mountain system proper of early Tertiary age lies to the east of the Selkirk and Purcell ranges and is the youngest mountain range in Canada. The Selkirk and Purcell Mountain ranges consist dominantly of sedimentary rocks for the most part of Pre-Cambrian (Beltian) age, intruded by granitic rocks, the latter being for the most part of Jurassic age. The Jurassic granites are cut by granites alkaline in character which can be referred to the mid-Tertiary period. The most notable example of these Jurassic granite masses is the West Kootenay batholith (the Nelson granite) whose eastern contact with the sedimentaries forms the western border of Ainsworth. The sedimentary rocks which occur around the granite batholith are, for a certain distance from the igneous contact, highly altered and in places impregnated with economic deposits of the valuable metals. Ainsworth district, as well as the Slocan district, occurs in this mineralized zone being underlain almost en-

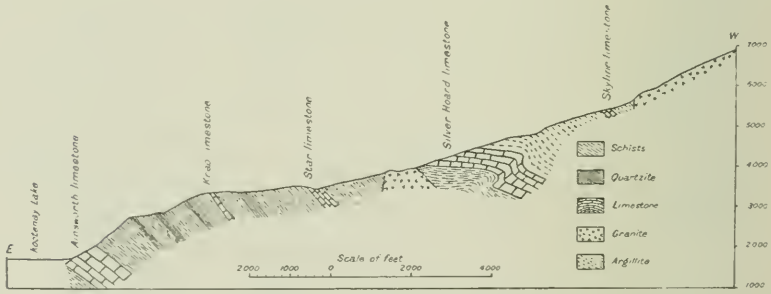


Fig. 2.—Structural Section through Ainsworth.

[1]S. J. Schofield. Science Conspectus. Vol. 4; 1914, p. 122.
S. J. Schofield. Scientific American. Vol. 78; 1915; p. 88.
[2]R. A. Daly. Geog. Jour. Vol. 27: 1906. p. 586.

tirely by altered sedimentary rocks. The sedimentary series consists of a variety of mica and hornblende schists with numerous interbands of limestone, quartzite, and argillite all striking approximately north and south and dipping on an average of 45° to the west. Intruded into the sedimentary series are small cross-cutting bodies of granite and numerous basic and aplitic dykes. The relationships of the rocks are shown in the accompanying cross-section of the district (Figure 2).

ECONOMIC GEOLOGY

The deposits at Ainsworth are entirely silver-lead; the chief mineral being silver-bearing galena except in the case of the No. 1 mine, the present workings of which expose only the oxidized ore zone of an original sulphide deposit. For the purposes of description only type deposits will be mentioned and for convenience they may be classified according to form as follows:

1.—*True fissure veins.*

(a) Cutting the bedding planes at an angle. Highland, Florence, Early Bird, etc.

(b) Parallel with the bedding planes. Maestro, Banker, etc.

2. *Replacement deposits in limestone.*

No. 1, Silver Hoard.

DESCRIPTION OF DEPOSITS

Highland.—The Highland group of claims occurs on Cedar creek, one and one half miles from Ainsworth with which it is

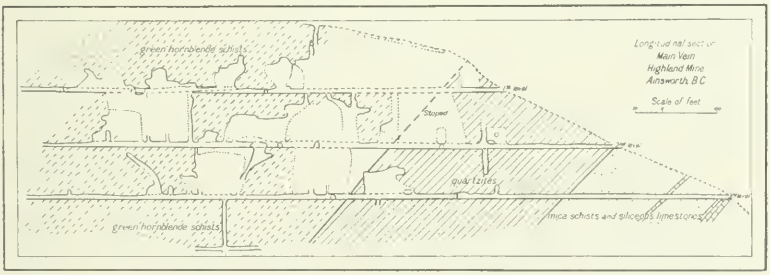


Fig. 3.—Longitudinal section along the main vein of the Highland Mine.

connected by a waggon road. An aerial train carries the ore from the mine to the mill on Kootenay lake at the mouth of Cedar creek. The country rocks of the Highland group consist mainly of siliceous limestones, quartzites, green schists, and quartzites in ascending order, (see Figure 3). This series is cut by mica and non mica basic dykes generally parallel with the bedding of the sedimentaries. The dykes vary in width from a few inches up to fifteen feet and are excellent horizon markers in determining the throw along the fissures. The deposits occur in true fissures, three in number trending approximately north-west with a dip of 75° E. The fissures are well defined and show a horizontal displacement of the walls, of 20 to 130 feet. In the quartzites the fissures are clean and generally have a filling of quartz, while in the hornblende schists the walls of the fissures have been badly decomposed for a considerable distance from the fissure itself. The ore consists mainly of coarse grained galena and zinc blende with smaller amounts of pyrite and chalcopyrite in a gangue of quartz. Ankerite and fluorite occur in cracks in the ore and are the last primary minerals to be deposited in the veins. The distribution of the ore bodies in the veins bears a striking relation to the quartzite-hornblende schist contact. From an examination of the accompanying section of the Highland mine (Figure 3) it can be seen that the ore bodies occur as tabular masses both in the quartzites and the hornblende schists and are as the upper contact of the quartzite with the schist. As this contact dips about 45° to the west the ore bodies pitch in the same direction.

Florence Mining Co.—The claims operated by the Florence Mining Company are located on Princess creek about one-and-one-half miles northwest of Ainsworth.

The country rocks on these claims are all sedimentary, consisting of interbanded siliceous limestones and mica schists striking north and south with a dip of 45° to the west.

The vein which cuts the above sedimentary series, strikes north-west and dips 75° south-west, and varies in width from a few inches to 16 feet. This variation in width is due to the character of the country rocks crossed by the fissures. In the mica schists the vein is barren but where it crosses the siliceous

limestone, a great enlargement takes place which is of the nature of a replacement deposit. Locally, these enlargements are called cross veins. The ore consists mainly of coarse grained galena with subsidiary amounts of zinc blende and iron pyrites. The gangue is mostly quartz and silicified limestone. The future development of this promising property should be based on the above facts which show that although the vein pinches in the schists, it is good prospecting to follow closely the main fissure in order to find other intersections of the vein with limestone bands.

Maestro.—The Maestro operated under lease by the Consolidated Mining and Smelting Company, occurs about 3 miles southwest of Ainsworth at an elevation of 1190 feet above Kootenay Lake. The sedimentary series in the vicinity of the Maestro consists of alternate belts of quartzites and green hornblende schists striking about north and south, and dipping on an average of 45° to the west. The vein is of the true fissure type with a strike and dip concordant with the surrounding country rocks, and can be traced both north and south through several adjoining claims. It is entirely enclosed in the green schists near the contact with a belt of underlying thin bedded platy quartzites, and varies in width from 6 to 8 feet. The ore consists of coarse grained galena with very little zinc blende in a gangue of quartz.

Silver Hoard.—The Silver Hoard mine is situated 7 miles by waggon road in a northwesterly direction from Ainsworth and at an elevation of 4300 feet above sea level, or 2540 feet above Kootenay lake. The ore occurs in two distinct zones in the Silver Hoard limestone along its contact with the overlying slate. The upper zone occurs at the contact with the slates, while the lower zone occupies a position from a few feet to 20 feet from the upper contact. The ore consists of galena, zinc blende and native silver with a gangue of quartz, calcite, and fluorite replacing limestone. No oxidized ore zone exists. The structure of the Silver Hoard is identical with that of the No. 1 mine (see Fig. IV.). The ore zones from the surface to the 100-foot level dip to the west. Here a shallow syncline occurs which passes into a gentle anticline. Between

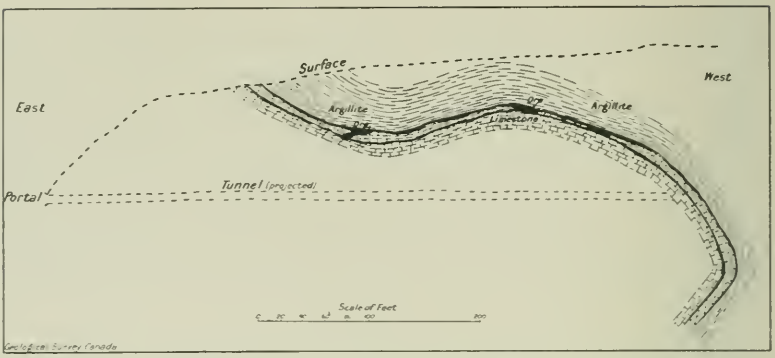


Fig. IV.—Diagrammatic cross-section through No. 1 mine, Ainsworth.

the 100-foot level and the 200-foot level, the ore zones change from a westerly dip to an easterly dip. Below the 200 foot level no information as to the structure is available, but it is probable the ore zones will resume their westerly dip at no great depth since the ore zones follow the folding of the rocks which, on the whole, dip at an angle of 45° to the west.

No. 1. Mine.—No. 1 mine is located 6 miles from Ainsworth at an elevation of 4200 feet above sea level or 2440 feet above Kootenay lake. From the accompanying diagrammatic cross section (Figure IV) it can be seen that the ore bodies occur as replacements deposit in limestone near its upper contact with argillite. The ore zones are two in number, one along the contact and the other from 7 to 20 feet from the contact. The present development lies entirely in the oxidised zone in which the ore occurs mainly as soft brown oxide of iron impregnated with native silver and probably some carbonates and sulphates of lead. The structure of the No. 1 mine is very simple. The ore zones follow the bedding planes of the surrounding sediments and hence, as shown in the sketch (Figure IV) occur in the form of synclines and overturned folds. The point of the overturn which occurs between the first and second levels in the southern end of the property, plunges to the north so that in the northern part of the property the overturn is found between the second and third levels. This explains the fact that as one goes from the southern end of the property to

the northern end on the first level the vein dips to the east, then becomes vertical and, in the northern end, dips to the west. The vein on the third level dips to the east, but since the whole sedimentary series at Ainsworth dips to the west, it is very probable the limestone-argillite contact will resume its westerly dip at no great depth, and that if ore is still present, the sulphides of lead zinc and iron will take the place of the oxidized ore which is found in the present workings.

AGE OF THE DEPOSITS.

Age relations of the replacement deposits and the true fissure veins.—The relationship of these two types of deposits was seen on the Ayesha claim where the replacement deposit occurs at the upper contact of a limestone belt with a schist. The replacement deposit with a north and south strike was cut by two distinct fissures trending approximately east and west, and dividing the deposit into three distinct portions arranged *en echelon* since the portion north of each fault was stepped to the west. Ore occurs in each of the fissures especially in that portion which joined two of the replacement bodies. The fissures are identical in character with the fissures described in a previous paragraph on the Highland mine. The mineralogy of the fissure veins and the replacement deposits is identical even to the proportion of silver to lead in the galena. The above evidence supports the conclusion that the fissure veins and the replacement ore deposits are contemporaneous in their formation.

Age relations of the dykes and the ore deposits.—As mentioned in the description of the Highland mine, there is a horizontal displacement of 30-130 feet along the fissures. The basic lamprophyre dykes which were faulted by this movement were also highly altered in the vicinity of the fissures by the ore bearing solutions thus proving that the ore deposits are younger than the basic dykes.

Age of the deposits.—The determination of the age of the deposits is rather a difficult matter since the information at hand is very meager. It can be said with certainty only that

the deposits are younger than the basic dykes and older than the glacial deposits. From a general study of the region it is believed that the dykes are closely related in age with the intrusion of the main mass of the West Kootenay granite batholith (the Nelson granite) of Jurassic age.

ORIGIN OF THE ORES.

The ore-bearing solution emanating from the granite probably in the pneumatolytic stage of cooling (since fluorite is a common gangue mineral in all deposits), rose in the fissure and when they reached a favourable locality formed ore bodies, either in the fissures (as in the vicinity of the quartzite-hornblende schist contact of the Highland mine), or in the rocks cut by the fissure (as in the Florence mine where the mineral bearing solutions attacked and replaced the limestone bands). The ore in the No. 1 and the Silver Hoard deposits evidently spread latterly from such fissures since the ore formed a regular continuous horizon at the limestone schist contact even in the over-turned portions of the contact.

FUTURE OF THE CLAY PRODUCTS INDUSTRY IN EASTERN CANADA

By J. Keele.

Annual Meeting, Toronto, 1915

The beginning of the present decade of growth and progress in Canada found the native clayworking industry inadequate to supply the demand for building material. Consequently, such substitutes as concrete construction, sand lime and cement bricks, were largely used, and much face brick and fireproofing were imported from the United States. During the last 3 or 4 years several large plants have been erected in the vicinity of Toronto and Montreal for the manufacture of ordinary building brick. The output of these plants is now so considerable that there is no longer an excuse to use substitutes for brick construction, either on the score of price, quality, or promptness of delivery. As competition in this field is bound to be keen, the future existence of the individual industries of this class will depend on their ability to produce the largest percentage of No. 1 bricks per kiln, with the lowest cost of fuel and labour.

The recent great development of the industry in the area between Toronto and Hamilton was due to two conditions: The district contains an abundance of the best raw materials in Canada for the manufacture of rough clay products; the two cities and neighbouring towns offer a large market for the wares produced from them. The raw material used includes plastic shales from two formations, the Queenston and Lorraine, which are better known to clay workers by their former names, Medina and Hudson River. The soft stoneless clays of recent origin are also more extensive in these localities than anywhere else in Ontario, except possibly in the clay belt of the northern portion of the Province.

The Lorraine shale and stoneless brick clays of recent origin occur abundantly within easy reach of the city of Montreal, and these have been utilized freely in recent years in building up that city and its suburbs.

The city of Quebec is supplied with similar materials, conveniently situated. The first plant for the manufacture of shale brick was established near this city a few years ago. The products of this plant have been so satisfactory in every respect that a bright future is assured for it.

The city of Ottawa was dependent until recently upon the common brick made for surface clays in its vicinity. A few years ago an extensive plant was erected at Russell, about 20 miles distant, which utilized the Medina shale outcropping in that neighbourhood, and the greater part of the output goes to Ottawa. The Lorraine shales outcrop in a large area just east of the city limits, awaiting future developments, but is not utilized at present. The Maritime Provinces possess the raw materials suitable for the manufacture of a greater range of clay products than is found in any other portion of Eastern Canada, but the surface clays are almost the only portion of their resources at present utilized. The principal brickmaking centres are at St. John, Pugwash and Annapolis, these points being selected on account of the raw material available, and the facilities for distribution both by land and water. The large plants which were recently erected in these localities use the stoneless surface clays for brickmaking, while all the large plants erected in the Provinces of Ontario and Quebec use either of the two shale formations before mentioned. Owing to their greater compressive strength, density and uniformity of size the shale brick are preferred for underground work such as trunk sewers, and for high structures carrying heavy floor loads. The future development of the brick industry in the Maritime Provinces will be in the utilization of the plastic shales with which they are bountifully supplied. As far as the rough clay products industry in rural districts is concerned, there will be little or no change in the future as there has been in the past. The character of the product is fixed by tradition and usage, and by the raw material available at any given locality. Any changes that are introduced will not affect the colour and shape of the product, but will be manifested in improved or more economic methods of production, such as the substitution of permanent down draft kilns for the wasteful temporary scove or clamp kilns in burning.

One thing seems bound to happen, however, in this connection, and that is the extension of the practice of underground drainage for agricultural lands. This is done to a considerable extent already in southern Ontario, but scarcely at all in Quebec or New Brunswick. No field drain tile are at present produced in these two provinces, although they have extensive and widespread deposits of stoneless, surface clays suitable for the purpose, as well as large regions of land which would be rendered much more productive by underdraining.

The development, therefore which is likely to occur, is the building of several small plants at various points, to which farmers from the surrounding districts can send their teams for a load of tile whenever required. This seems to be a more satisfactory plan than in having a few extensive tile works at certain centres, from which shipments would have to be made by rail or barge to the rural districts.

The extensive belt of stoneless glacial clays in northern Ontario and Quebec contains material which is very suitable for the manufacture of common brick and drain tile. These localities are situated too far from markets at present to allow of them being worked for this purpose. Owing to the abundance of wood in this region, brick and tile should be produced cheaply. A cheap freight rate to southern points might be secured by utilizing returning empty trucks which convey coal to the divisional points on the Transcontinental railway, which traverses the clay belt.

Face Brick

The discussion so far has reference to common brick made by either the soft mud, wire cut, or dry-pressed process. So much poor dry-pressed brick has been made in Eastern Canada in late years, that I offer no apology for placing such in the common class. They can be bought at common prices also. Some of the makers do keep up a high standard of excellence as regards density, finish and colour for their dry-pressed product, and these always find a ready sale at prices considerably above that of the ordinary building brick. The best practitioners in architecture, however, no longer regard the dry-pressed process as the high

water mark in brick production. The smooth surfaces and tiresome uniformity of colour do not lend themselves to effective treatment, especially for domestic work; but it is probable that their use will be continued for business buildings. Roughness in texture and variety of colour are now regarded as most desirable qualities for face brick. The roughness is produced by a simple device attached to the brick machine, and variety of colour is obtained by 'flashing' during the burning process. In some localities in the United States kilns of this kind of brick are turned out which rival Turkish rugs in colour and texture. These bricks are made from shales, the cost of production not being much more than for ordinary wire-cut; but, the profits are exceedingly greater.

A few of the leading brick plants in Ontario and Quebec have taken up the manufacture of rough faced brick n a half-hearted manner as a recognition to a passing fad. These firms are essentially dry-pressed face brick makers; but, it is significant that by overburning and flashing their dry press wares they are catering to the desire for roughness of finish and diversity of colour. Of the twenty plants using either the Queenston or Lorraine shales in the triangular area included between Toronto, Hamilton and Credit Forks, twelve are wholly dry press makers, while eight are wire-cut or mixed plants. Three of the latter make a small quantity of rough texture face brick. One of these, the Ontario Government plant at Mimico, produces an article equally as good as most of the imported material. There appears to be an opportunity here for an enterprising brickmaker to produce a distinctive article which will bring him out of the common class into the region of greater profit.

One of the plants at Laprairie, near Montreal, uses the Lorraine shale to good effect in producing a rough texture face brick for the Montreal market.

The Lorraine and Levis shales near the city of Quebec can be used to produce richer effects than those of the western shales.

The carboniferous shales in New Brunswick and Nova Scotia contain endless possibilities for this class of wares. That they have been unused up to the present, denotes a singular lack of appreciation for the beauty of burned clay ware, and an unrivalled building material, in the cities of these Provinces.

Fireproofing

Hollow blocks of various shapes made of burned clay are used largely as the fireproof clothing for steel frames in modern high structures; hence the trade name of fireproofing for this class of ware. The use of hollow blocks, however, is not confined to covering steel structures; but they are now used extensively as a building material for the construction of exterior walls and partitions in factories, silos, dwellings and storehouses. There are only four factories manufacturing this class of ware in Eastern Canada, and only one of these is devoted solely to this product. Three of these use surface clay, and one located in Toronto uses Lorraine shale.

The surface clay is not a very satisfactory material for fire proofing, the shales being better, as they produce a harder, stronger ware, which also stands transportation well without undue loss from breakage. The supply of home-made fireproofing is inadequate in quality and quantity. Several brick plants already located on shales in any of the provinces might profitably take up this branch of the industry. Almost any of the plastic shales between Lake Huron and the Atlantic are suitable for the purpose. The use of hollow blocks made from burned clay is always increasing, and will continue to do so as long as building activity keeps up.

Paving Brick

Vitrifying clays or shales suitable for the manufacture of this class of clay products are comparatively rare in Eastern Canada. Nearly all the paving brick used in Toronto and other localities in southwestern Ontario are imported from the United States. A small quantity of paving brick is made from the Lorraine shale at West Toronto. While these may be suitable for sidewalks or residential streets, their life is too short under the ordinary city traffic.

Certain shale deposits of the Levis and Sillery formations, which are probably suitable for the manufacture of paving brick, were discovered by the writer about two years ago. The Citadel Brick Company of Quebec is experimenting with these

shales at present, so that paving brick may become one of their principal products in the future.

It is quite possible that paving blocks might be made from the Lorraine shales near Toronto or Montreal if a certain percentage of imported fire clay was added, in order to prolong the vitrification range of the shale and toughen the product. This is another opportunity for brickmakers in Ontario to get out of the common brick class.

The Carboniferous formation in the coal mining districts of New Brunswick and Nova Scotia contain shale beds suitable for the manufacture of vitrified wares. No attempt has yet been made to develop this industry, the principal reason being that a market does not exist for them at that end of the country.

Sewer-Pipe

Three plants are employed all the year round in the manufacture of sewer-pipe in Ontario. The raw material used is either the soft weathered top of the Queenston (Medina) shale, or a transported clay washed from the same material, which occurs at Waterdown and vicinity, near Hamilton. This clay burns to a good strong red body; but does not take a very good salt glaze. It is the only material we know of in the province which is suitable for the purpose. These pipes are used extensively in southern Ontario, notwithstanding the competition of the brightly glazed imported pipe made from fire clay or semi-refractory clay in Ohio or Missouri. The most extensive sewer-pipe plant in the Dominion is situated at St. John in the Province of Quebec. The local clay is used; but a liberal stiffening of fire clay imported from New Jersey is added to it. The products of this plant are very satisfactory in every respect, and have a wide distribution.

The shales already indicated as occurring in the vicinity of Levis might be used to advantage for this c ass of ware, as they take a good salt glaze, and are by far the most refractory shales found in the two provinces. With water transport on the St. Lawrence and rail connection with all the principal centres, an important industry in vitrified wares might be establ'shed at Levis.

The Carboniferous shales of the Maritime Provinces furnish the necessary raw materials for the manufacture of sewer-pipe, as well as the coal for burning them. One plant has been established for several years at New Glasgow in Nova Scotia, and is now in a flourishing condition.

There is room for another sewer-pipe plant at Moncton or in the Grand Lake coal area of New Brunswick. Ware like this is not like common brick, inasmuch as it can stand the freight charges for long haulage by rail.

Electrical Conduits

These are used for laying wires underground, their requisites being strength, straightness and impermeability. Conduits are generally made from fire clays or some semi-refractory clays or shales, finished with a salt glaze. None is produced in Canada; but hundreds of thousands of dollars' worth are imported annually from the United States. The telephone and telegraph companies prefer to go to the additional expense of conduits because of the saving in cost of maintenance and the superior service they give. Raw materials for this purpose are available in the Maritime Provinces alone. There is now a sufficient market in Canada to keep a large plant running on conduits alone, or it might be combined with the manufacture of sewer-pipe and silo blocks.

Architectural Terra Cotta

None of this material is produced in Canada; but it is now used extensively in all our cities. It is general y made with a fire clay or a buff burning, semi-refractory clay, the finished ware being coated with a glaze or enamel, either white or tinted in colour, or mottled to imitate granite. An industry of this kind requires a large staff of skilled workmen, for whom employment must be found all the year round. Now that the Canadian market for architectural terra cotta is large, a plant situa ed, for instance, on the Welland Canal, where fire clay and coal could be brought by barge, ought to succeed. The manufacture of enamelled brick and decorative tile for indoor work could be combined with the terra cotta, as the same body and enamels could be used

for all. The manufacture of red and buff terra cotta, without a glaze, was carried on at the Don Valley Brick Works in Toronto some years ago; but it was discontinued. This industry might be revived again to advantage. Cornices, string courses and panels of this material could be combined with brickwork and made very effective in many classes of buildings. The Lorrain shales, the weathered top of the Queenston shales and certain of the surface clays can be used for this purpose.

Roofing Tile

The manufacture of roofing tile is not carried on in any part of Canada on a commercial scale. A few day workers in eastern Canada have made tile for roofing houses in the vicinity of their plants; but none care to undertake making them for sale. The manufacture of roofing tile at the Ontario Government plant at Mimico has passed the experimental stage. They are now producing red tiles from the Lorraine shale which is as good in colour and body as any of the imported tile. These tiles are for the new Asylum Buildings at Whitby; they are of the Spanish or S type. A large quantity of roofing tile is imported annually into this country for use on the best class of buildings. The shales in Ontario or the Maritime Provinces will supply the raw material; but a certain amount of skill is required to make them.

Fuel

Fuel is one of the most important items to be considered in the future of the clayworking industry of Canada. Not so many years ago wood was the only fuel used for burning. It was plentiful and cheap, and no very high temperatures were required for the class of clay wares then being made. Wood is increasing in price, and harder to obtain in large quantities; higher temperatures with longer sustained fires are required for the better class of clay wares; so that more coal than wood is now used in the clayworking industry in this country.

No coal occurs in either the Provinces of Ontario or Quebec, and very little in New Brunswick. Nova Scotia coal is shipped as far as Montreal; very little of it comes further west. Most of the

coal used in Ontario comes from Pennsylvania. There is great waste of coal in the clay industry, due to defective or poorly built kilns, badly constructed furnaces giving imperfect combustion, careless firing and other causes. The use of producer gas or a perfect combustion furnace, are two methods proposed by the brickwork engineer for the elimination of waste in fuel. A discussion of the various devices in kilns and furnaces is beyond the scope of this paper; but it has been demonstrated beyond question in a few plants that an immense saving of fuel can be effected and the smoke nuisance almost eliminated.

Refractory Goods

Clays suitable for the manufacture of refractory or even semi-refractory wares do not occur, practically speaking, in the provinces of Ontario and Quebec, as far as we know at present. Fire clay is known to exist at one point in Ontario, but is situated too far from transportation to be of economic value. This occurrence is situated on the Missinabi River about 40 miles north of Cochrane, a point on the National Transcontinental Railway. The deposit consists of white, pink and yellow plastic clays, the white clay only being a fire clay. Borings made on the deposit show at least 25 feet in thickness over a considerable area, 15 feet of which is white clay.

A four-foot bed of semi-refractory clay, softening at a temperature of about 1600° C., is found underlying the coal seam in certain portions of the Grand Lake Coal area of New Brunswick.

Plastic fire clays occur at three localities in Nova Scotia, at Shubenacadie, Inverness and Middle Musquodoboit. There is no railroad connection near the latter deposit. A large body of felsite, which in this instance is a refractory rock but not plastic, occurs at Coxheath near Sydney. When this material is crushed and mixed with enough plastic fire clay to make it workable, an excellent quality of fire brick can be produced. No fire brick is made from the native refractory clays in the Maritime Provinces. One of the principal reasons for this is that the Scotch fire brick can be delivered from steamships at a very reasonable rate.

Stove linings and special shapes of fire clay goods are made at St. John and Montreal in the Province of Quebec. The fire clays for this purpose are imported from the State of New Jersey.

Silica brick are made from crushed quartzite, to which is added about 3 per cent of lime for bonding purposes. These bricks are moulded by hand, and burned in kilns at a very high temperature. The raw material for this class of refractory ware occurs in Canada; but the cost of fuel for burning is large.

Whiteware

This term is used here to include table ware, sanitary and electrical porcelain, white, white floor and wall tile. The raw materials entering into the composition of all whiteware bodies are:—(1) kaolin, which is generally known by the trade name of china clay; (2) ball clay; (3) feldspar; (4) flint.

A limited amount of china clay is produced in Quebec, at a point about 40 miles northwest of Montreal. This is the only locality so far known at which it occurs. No ball clay has so far been found in Canada. Canadian feldspar is mined north of Kingston, Ontario, and shipped to New York State for grinding. Flint is generally ground quartz in America. Most of it is produced in the State of Maine, where it is ground for use in potteries. A large amount of fire clay is also used in the pottery industry for making saggars, which are practically fire clay boxes for holding the ware while burning. Whiteware industries are often situated at points where none of the raw materials occur, except possibly the fire clay. Convenience of transportation, a cheap fuel, and a large supply of skilled labour are the essentia features of a pottery centre.

No table ware is made in Canada. There are two plants at St. John, Que., which manufacture sanitary porcelain.

Electrical porcelain is now made in Hamilton, Ontario, by the Canadian Porcelain Company. Their output is chiefly large insulators for power lines. The General Electric Company also manufacture a certain amount of their porcelain at Peterboro. A plant for the manufacture of white vitrified floor tile was recently built at Kingston.

Every particle of the raw materials used in the above industries is imported from England and the United States. As the consumption increases these factories will be enlarged or probably other factories will be established at various points in the future. Economic considerations are probably against the establishment of an industry for the manufacture of table ware in Canada. These articles will continue to be imported for a considerable time to come. The only difference will be that a larger proportion will come from England, France and the United States, which hitherto came from Germany and Austria.

Requirements of the Industry

The time is past when anything that looked like a brick could be sold at a profit. The future demands will be more exacting, so that those clay working plants which do not strive after excellence in production or are located on poor material, will have to drop out of the business. The weak spot in the clay-working industry is the lack of technical knowledge on the part of the majority concerned in it. Many costly mistakes are made every year in various parts of the country, due either to poor location, wrong equipment, or defective raw material. Considerable skill and experience is required to build a modern clay-working plant and start it into successful operation; but, the tendency seems to be to employ the cheapest advice, on the popular assumption that anyone can make brick or tile. Clays contain more disagreeable surprises for unwary manufacturers than any other kind of raw material; but it is often possible to detect and provide against these troubles in the preliminary examination.

The raw materials used in whiteware industries are fixed, and to a certain extent, unvarying. It is possible to foretell very closely what kind of ware will be produced in practice.

The points to be considered in starting an industry of this kind are: (1) A market for the wares; (2) location of plant; (3) fuel supply; (4) conditions favourable to permanent supply of skilled labour. These questions of course are dependent on the vital one, whether foreign competition can be met. In the

manufacture of clay products for structural materials where the raw material is of an infinite variety, no reasonable trouble or expense should be spared during the preliminary investigations for the determination of the proposed site.

EXTENT OF THE CLAY PRODUCTS INDUSTRY IN EASTERN CANADA

The following figures compiled by Mr. John McLeish of the Bureau of Mineral Resources and Statistics of the Department of Mines, shows a considerable annual increase in the output of clay products, during the last three years.

VALUE OF PRODUCTION OF CLAY PRODUCTS BY PROVINCES

	1911	1912	1913
Nova Scotia	$ 274,249	$ 272,053	$ 332,272
New Brunswick	38,000	54,910	62,269
Quebec	1,341,467	1,680,460	1,606,816
Ontario	3,916,575	4,864,700	5,220,467
Total value for Eastern Canada	$5,570,291	$6,872,123	$7,221,824

EXTENT OF PRODUCTION AND CONSUMPTION OF CLAY PRODUCTS IN THE DOMINION

Total value of Clay Products produced in Canada during 1913	$9,504,314
Total value of Clay Products imported into Canada during 1913	7,237,885

Owing to the depression in the building trade during the latter part of the season of 1914, it is probable that some decrease will be shown in production of rough clay products when the returns are compiled. The clay industry, however, does not suffer to the extent that many others do in times of general depression.

ORIGIN OF THE WABANA IRON ORE*

ALBERT O. HAYES, Ottawa, Ont.

Annual Meeting, Toronto, 1915

The conclusions regarding the origin of the Wabana iron ore presented in this paper have been reached as a result of field and laboratory work carried out under the auspices of the Department of Geology of Princeton University, and the Nova Scotia Steel and Coal Company. I have been aided in many ways by the management of the Nova Scotia Steel and Coal Company and desire to acknowledge, especially, the kindness of Messrs. Thomas Cantley, R. E. Chambers, A. R. Chambers, and C. L. Cantley, and to thank the company most heartily for their courteous co-operation.

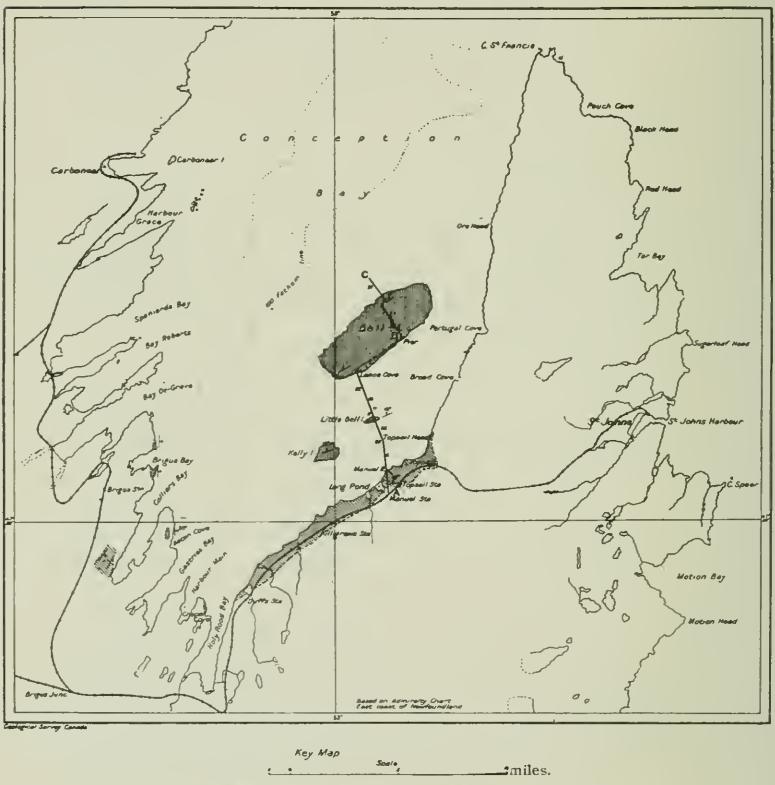
It gives me great pleasure, also, to express my deep gratitude to Professors C. H. Smyth, jr., and Gilbert van Ingen, of the Department of Geology, Princeton University, for their continued and generous support throughout this investigation Professor van Ingen has also identified the fossils referred to in the text and prepared the photographs.

Certain names have been given to the various iron ore beds, and among others "Dominion bed" and "Scotia bed" to the principal workable beds. These names were applied before submarine mining commenced when the Dominion bed was operated by the Dominion Iron and Steel Company, and the Scotia bed by the Nova Scotia Steel and Coal Company, on the land claims on Bell island. In the case of the submarine holdings, however, each company owns all of the beds within its respective areas, the boundaries to which are vertical. At the present time the Scotia Company is mining largely from the Dominion bed as well as the Scotia bed, while the Dominion Company owns a part of the Scotia bed as well as part of the Dominion and other beds in their submarine property. Hence the names of the ore beds do not indicate the names of the owners.

* Published by permission of the Geological Survey of Canada.

The difficult pioneer work of developing the property for this company was ably managed by Mr. R. E. Chambers, and we are indebted to him (7)* for the first description of the deposit, published in 1896.

An historical sketch and description of the development of mining operations were given in 1909 by Messrs. R. E. and A. R. Chambers (8), and in 1911 an article by Mr. Thomas Cantley (3) reviewed the history of the mines, and described the ore beds and methods of mining.



Key Map of Southeastern Newfoundland.

* The numbers in brackets refer to bibliographic list on page 245.

General Stratigraphy of the Ore Zones

The Wabana iron ore forms part of a series of Ordovician sediments which are exposed on Bell island, in the south central part of Conception bay. The ore beds outcrop for about three miles along the northern shore of this island and dip to the northwest. They are the highest beds of the northern limb of an anticline, the truncation of which has revealed an almost continuous series of unmetamorphosed sandstone and shale of several thousands feet thick underlying the ore, and ranging in age from lower Cambrian to lower Ordovician.

An extraordinary concentration of ferruginous minerals occurs in five principal zones through the upper 1000 feet of strata, which comprise, with the accompanying sandstone and shale, a fossiliferous series of shallow water deposits. The conclusion has been reached that the ore beds are primary sediments which have been deposited on the floor of a marine sea of Arenig to Llandeilo age, and their distribution regulated by the same forces which have had to do with the formation of the sandstones and shales which accompany them.

The strata are exposed at the base of the series at Manuels brook on the south shore of Conception bay. A little more than one-half the distance across the strike of the strata is covered by the water of Conception bay, but the exposures on Kelly and Little Bell islands furnish a section considerable over 1000 feet in thickness at a point about midway across the covered portion. This section reveals strata of a character very similar to that found on Bell island. The sediments are laid down in broad lenticular beds of varying thickness and extent, which occasionally begin at a point, spread and thicken and then die away within a distance of 100 feet, although usually they are much more continuous. Cross-bedding and ripple marks are constantly met with. It is evident that these sediments were deposited on a sea bottom that suffered continual oscillation relative to the surface of the water. Strata varying in thickness up to about 100 feet, composed predominantly of sandstone or of shale, are very persistent and can usually be traced as far as exposed above water. The individual layers constituting these zones are

themselves continually thickening and thinning, being replaced by others of similar character. The ore beds display all the features peculiar to off-shore deposits but were accumulated generally in thicker and more extensive individual layers than those of the accompanying rocks, and the thicker beds of oolitic iron ore appear to be the most persistent of the series.

Oolitic iron ore and ferruginous rocks have been found in six zones on Bell island and these are given in stratigraphical order, beginning with the lowest.

Zone 0.—A thin ferruginous band occurs in the strata forming the extreme southwest part of the Island, outcropping from Lance Cove westwards to the western shore near the "Clapper".

Zone 1.—This zone is partially exposed by shallow cuttings along the tramways of the Nova Scotia Steel and Coal Company, and the Dominion Iron and Steel Company near

Fig. 1.—Dominion bed, surface stripping.

their intersection at Kent's Bridge. It may be traced by intermittent outcrops from Eastern Head on the east coast of the Island to the northwest coast between Big Head and the Bell. It contains bands of oolitic hematite, two of which appear to be continuous, one attaining a thickness of about two feet.

Zone 2, including the Dominion bed.—The lowest oolitic hematite of this zone is about 600 feet stratigraphically above zone 1, and the whole zone consists of a series of bands of oolitic hematite alternating with shales and cross-bedded, fine-grained sandstones, comprising about 100 feet of strata culminating in the Dominion bed. The top 35 feet of strata are composed largely of oolitic hematite in thick beds with thin ferruginous sandstone and shale parting rocks (Figure 1).

Zone 3, Pyrite layers.—Oolitic pyrite occurs in from one to three bands extending through strata from a few inches up to four feet in thickness, and found from one to ten feet above the highest oolitic hematite of Zone 2.

Zone 4, Scotia bed.—This zone commences 210 feet above Zone 3 and is confined to about 15 feet of strata.

Zone 5, including the Upper bed.—The lowest band of this zone is separated from the top of the Scotia bed by about 40 feet of sandstone and shales, and the oolitic iron ore continues upwards through about 50 feet of strata.

Palaeontology.

A number of distinct faunal horizons occur in the ore measures which may be roughly correlated with the ore zones. No accurate descriptions of the range of the faunas is here attempted. In the ferruginous sandstones and oolitic hematite beds of zone 1, the following fauna has been found: *Lingula affinis, Lingulella bella, Sphaerobolus fimbriatus, Obulus burrowsi, Cruziana similis, Arenicolites and Sphaerobolus spissus.*

In the oolitic hematite of the Dominion bed, zone 2, *Lingula hawkei* occurs, and boring algae are abundant. In the upper part of Zone 2 above the Dominion bed, a pebble bed

carries a Helmersenia-like brachiopod. This species is absent from the overlying shales of zone 3.

Above a disconformity at the top of zone 2, the pyrite beds with the accompanying and overlying shales of zone 3, hold *Didymograptus cf. nitidus*, abundantly. Thin shelled *orthoceratites* are also found, and *linguloid* brachiopods occur in the pyrite layers.

From the oolitic ore of the Scotia bed and accompanying rocks of zone 4, the following fauna has been taken: Trilobites including an asaphid, probably *Niobe*, and a Procalymene, *Schizocrania new sp.*, *Lingula new sp.*, *Hymenocaris* (?), tail of a Eurypteroid crustacean, *Cruziana similis*, *Arenicolites*, *Eophytum linneanum*. Boring algae are abundant in the iron ore.

Zone 5 holds the following forms: *Schizocrania*, *Lingula leseueri*, *Lingula sp.*, *Orthis sp.*, *undet.*

The trails known as *Cruziana similis*, *Arenicolites*, *Eophytum linneanum* are found throughout all zones. Raindrop impressions are found at locality 206 A17 between zones 1 and 2. The *Schizocrania* occurring in the pebble bed 206 J3 of zone 4, and 206 AH section of zone 5, exhibit a peculiar feature. The laminae of the shells of these brachiopods, and of *Lingula*, frequently have a thin coating of galena over their surfaces, giving them a shiny appearance. This remarkable association of lead sulphide and fossil shells is worthy of further study. Small concentrations occur in the upper part of the Scotia bed and may be genetically associated with these fossils.

The general bearing of the presence of this fauna in the ore rocks and the especial significance of the incursion of graptolites in zone 3 will be discussed under 'Origin of the Ore.'

PETROLOGY AND CHEMISTRY

The ore is everywhere oolitic, the individual spherules varying in size from .1 to .5 millimeters in diameter. Under the microscope the spherules are seen to consist of alternating concentric layers of hematite and chamosite. Siderite becomes

locally abundant as a cementing material. In dense ore the opacity of the hematite frequently masks the structure of the spherules, and they appear to be composed entirely of iron oxide. By digesting such dense ore in hydrochloric acid, however, while the iron oxide dissolves, the spherules retain their original size, become white and translucent and consist of concentric layers of a material readily soluble in fixed alkalis, exhibiting the characteristics of amorphous silica. A similar experiment was made on ore the spherules of which consisted of chamosite free from hematite, and similar skeleton spherules of amorphous silica obtained. It seems evident, therefore, that the skeletons result from the decomposition of chamosite which is present in the spherules even when entirely masked by the more opaque hematite. These experiments were suggested by Prof. Smyth (11: pp. 487-496) who obtained like results by a similar treatment of the oolitic Clinton ores (Figures 2, 3, 4).

Fig. 2.—Oolitic iron ore from Scotia bed.

Fig. 3.—Photomicrograph of thin section of iron ore from Scotia bed. Magnification 110 diams. Slide 23. Neg. 1351.

a.—Spherule composed of concentric layers of chamosite and hematite.

b.—Mat of boring algae in upper part of phosphate nodule.

c.—Spherule composed of concentric layers of chamosite with a nucleus of calcium phosphate-like material holding boring algae.

d.—Matrix of iron carbonate.

e.—Chamosite spherule with quartz nucleus. The outer portion of chamosite is replaced by siderite.

A chemical analysis of oolitic chamosite from the upper part of the Scotia bed gives:

SiO_2	25.64	per cent
Al_2O_3	19.75	"
FeO	39.74	"
MgO	2.98	"
H_2O	11.89	"

The mineral is green in colour being essentially a hydrated silicate of ferrous iron and alumina.

The fossil lingula shell fragments have a composition of between 60 and 70% calcium phosphate, and the lime and phos-

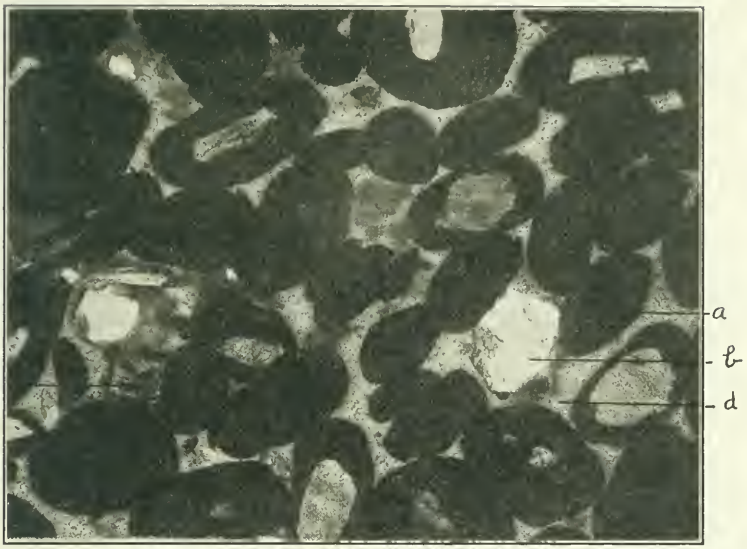


Figure 4.—Photomicrograph of thin section of iron ore from Scotia bed. Mag. 110 diams. Slide 18. Neg. 1341.
a.—Spherule of dense hematite.
b.—Quartz fragment.
c.—Shell fragment.
d.—Chamosite matrix.

phoric acid present in the ore are derived principally from this source. The ores and accompanying rocks appear to be free from any original lime content in excess of that which is found in combination with phosphoric acid in the shell fragments.

The only minerals found to have been introduced secondarily to the iron are calcite and quartz, and these fill fault cracks which vary in size from microscopic veinlets to veins several inches in thickness.

All of the constituents of the ore vary within certain limits at different localities and portions of the ore beds, as shown in the following table:

Hematite	50 to 70	per cent
Chamosite	15 to 25	"
Siderite	0 to 50	"
Calcium phosphate	4 to 5	"
Calcite	0 to 1	"
Quartz	0 to 10	"

Zone 3 consists of layers of iron sulphide in the form of oolitic pyrite intercalated between beds of shale.

ORIGIN OF THE ORE

1. *Primary Nature of Ore Beds.*—The iron ore occurs in strata forming an integral part of a series consisting principally of shales and sandstones. The series is of Lower Ordovician age and since it exhibits closer relationships with European than with American geological formations, it is referred to the Arenig and Llandeilo of Wales, corresponding roughly with the Beekmantown, Chazy and Black River of the Appalachian province.

I have concluded that the iron ore occurs as a primary bedded deposit and that the iron content was present in the sediments at the time that the series was laid down. It is probable that the spherules were formed out of the extremely fine-grained, unconsolidated, ferruginous sediments of the sea bottom, in water sufficiently shallow to allow of a certain amount of agitation due to the action of the surface waves.

In the course of the deposition of a thickness of about 400 feet of strata in which the three workable beds are contained, records are found of a number of changes of sea level relative to the sea bottom and shore. Ripple marked surfaces in all of the workable beds are evidence of shifting of the bottom deposits by currents off a shore line. These are beautifully shown on the floors of the workable beds of zones 2 and 4. A disconformity occurs above zone 2, varying at different localities from a few inches to a few feet above the highest hematite of the Dominion bed, immediately above which the oolitic pyrite and black shales holding graptolites were laid down. The incursion of graptolites signifies a modification of the sea bottom such as to allow the passage of continuous ocean currents, capable of bringing in these planktonic organisms, which are considered to have been transported as are similar forms of life existing to-day in the Sargossa Sea, and it is thought that a considerable deepening of the sea took place at the close of the deposition of zone 2. Shallow water conditions again prevailed at the close of the deposition of zone 4, probably accompanied by considerable vertical oscillations, evidenced by a ripple marked

surface in the ore found two feet above the floor of the workable bed, and by the occurrence of a pebble bed, occupying erosion channels in the fine-grained sandstone, about one foot above the top of the Scotia ore. The ore above the erosion surface in the upper part of the ore bed is cross bedded, indicating the progressive accumulation of the spherules at the edge of a submerged bar, arranged in lamellae at a steep angle to the bedding plane.

The Scotia bed, where examined at the surface outcrop, contains a relatively small amount of detrital quartz fragments and has an exceptionally high siderite content. The ore zone is confined to about 15 feet of strata, about half of which thickness is composed of workable ore, occurring in a compact layer containing thin parting lenses of ferruginous sandstone at some localities, but entirely free from these layers over other areas.

2. *Conditions during Deposition.*—Considerable direct evidence is thus available on which to base a conception of the conditions obtaining during the deposition of the Scotia bed. The absence of detrital quartz in the ore indicates that the ore was formed at some distance from a shore line, while the ripple marked surfaces and cross bedding of the spherules required shallow water for their formation. The Upper bed also exhibits similar phenomena characteristic of such marginal deposits.

That the ore was deposited in marine waters is evidenced by their content of marine brachiopods and trilobites. The graptolites in the shales and oolitic pyrite also demonstrate the salt water origin of zone 3.

It is evident that the spherules were formed before the consolidation of the ore, since the ore is sometimes found to be made up of thin lamellae of oolitic hematite, and earthy hematite in alternating layers arranged by the sorting action of water. The spherules where closely packed together in the denser portions of the ore, are much flattened by vertical pressure and lie like tiny discs placed horizontally and where the spherules are disseminated more sparsely, they remain much more spherical. Spherules in certain layers parallel to the bedding of the ore are frequently much more flattened than others. This flattening probably occurred while the spherules were in a soft condition

when variations in the thickness and density of overlying sediments would result in differing amounts of crushing.

3. *Evidence from Fossils.*—Another most convincing fact is that worm burrows are found in all the ore zones, showing that the worms have burrowed from the erosion surface and have left cavities which became filled with an earthy material in which spherules rarely occur.

The fossil boring algae are minute tubular organisms found splendidly preserved in brachiopod fragments and spherules of hematite, and in chamosite and phosphate nodules. Similar boring organisms are known in modern seas and the presence of these fossil forms in both shells and spherules indicates that these materials were pierced while resting on the sea bottom. The tubules of these algae, the delicacy of which may be appreciated by stating their dimensions which vary from one fifth to four microns, often extend beyond the walls of the shells into the surrounding matrix. The portions of the ore in which these algae occur in this fashion could not have been subjected to much mechanical rearrangement. It is evident, therefore, that not only the spherules but the material which surrounds them must have formed a part of the unconsolidated sediments.

4. *Source of the Iron.*—Many questions present themselves regarding the source of the iron, and the various complex changes which it has passed through before reaching its present state. No direct evidence bearing on these questions is available and any statement must be necessarily theoretical.

F. Villain (14: pp. 1291-1293) proposed for the oolitic iron ores of Luxembourg and Lorraine a juvenile source consisting of outflows of iron-bearing solutions from fissures in the earth's crust, but no evidence whatever has been found on which to base such an hypothesis for the Wabana ore.

It seems probable that the iron was derived by long continued weathering of earlier crystalline and sedimentary rocks, the solution of their iron content by mineral and vegetable acids and subsequent transportation of the iron salts by streams into the sea.

Both the Pre-Cambrian sedimentary and crystalline rocks contain much iron and these have furnished the iron of the

Cambrian and Ordovician rocks underlying the Wabana ores. The Upper Cambrian rocks of Manuels brook and the higher strata outcropping on Kelly and Little Bell islands, contain a considerable amount of disseminated iron-bearing minerals including sulphide, silicate and carbonate. Occasional concentrations of one or more of these minerals occur. The Upper Cambrian beds of Manuels brook hold considerable iron sulphide and some carbonate. A ferruginous bed on Kelly island contains 27.75% of metallic iron mostly in a form resembling silicate with probably considerable carbonate. The shale at the Nova Scotia Company's pier on the south shore of Bell island holds 8.83% ferric oxide. The shales generally appear to be rich in iron, and as the greater part of this Cambro-Ordovician series is composed of shales, the whole series holds an unusually large amount of iron.

5. *Evidence from Absence of Limestone.*—There is a general absence of limestone from this Cambro-Ordovician series above the Middle Cambrian, and there is no evidence that any has ever existed, none having been found in the Upper Cambrian exposed on Manuels brook, or in the Ordovician measures on Kelly, Little Bell or Bell Islands. Chemical analyses of shale and sandstones underlying the ore beds on Bell island show an entire lack of lime, while many analyses of the beds and accompanying rocks prove the total content of about 2.5 per cent of lime occurs largely as a phosphate derived from fossil remains, and that little or no calcium carbonate occurs as an original constituent of the ore and accompanying rocks. The theory proposed by Professor L. Cayeux (4: pp. 284-285) for deposits of similar character and age in the Amoricain Peninsula, France, and suggested for all similar Palaeozoic deposits, that an original limestone has been transformed into an oolitic iron ore, is absolutely untenable for the Wabana ore, where every observation confirms the conclusion that the ore was formed originally in essentially the same condition excepting induration, in which it is found to-day.

A much longer period is represented by the Wabana ore series than would be required merely for the continuous deposition of such a thickness of rocks. About 200 feet below the Dominion bed the rocks were, when soft, exposed above water,

for raindrop impressions are here preserved pointing to an emergence shortly after their deposition. A change in fauna at the top of zone 2 occurs coincidently with a disconformity marked by an erosion surface, and a deepening of the sea followed. The Scotia and Upper beds also give evidence of many oscillations in the relative level of land and sea. Professor Gilbert van Ingen has suggested that successive marginal tilting probably occurred contemporaneously with the deposition of the rocks of Bell island, causing the shore line to advance and raising the underlying measures above sea level. These soft sediments with their large iron content, would then be available to further enrich the sea with iron solutions.

The chemistry of the processes involved in the formation of the ore must of necessity be very complex. Green iron silicates, resembling chamosite, are the most generally distributed primary iron-bearing constituents of both the ore and their accompanying rocks. Hematite is intimately associated with chamosite and is the most abundant iron-bearing mineral of the ore.

6. *Mode of Precipitation of Iron.*—Since W. Spring (12: pp. 47-62) has shown that ferric hydroxide will become dehydrated in salt water and form ferric oxide, the view may be taken that some of the iron may have been precipitated as ferric hydrate and formed the ferric oxide directly. If so, both the silicates and oxide have formed together in a most intricate fashion to form alternate layers in the spherules. It appears probable that a considerable proportion of the iron was precipitated primarily as ferrous aluminous silicates similar to chamosite and thuringite, and while a small amount of hematite appears to have been formed about tubules of boring algae, apparently secondary to the chamosite, most of the hematite is of contemporaneous origin with the silicates. The siderite was the last iron-bearing mineral to form and it frequently replaces chamosite, hematite and quartz. It may have been produced by the action of organic matter on chamosite and hematite during a period of shallow water conditions such as prevailed during the formation of the top of the Dominion bed, zone 2, and much of zone 5.

It has been shown that the ore of zones 2, 4, and 5, was formed as off-shore deposits on the bottom of a sea in which marine life was abundant. Some organisms, such as worms, actually burrowing into the oolitic ferruginous sediments. The question regarding the mode of precipitation of the iron on the bottom of an open sea, still remains unanswered. It is possible that during the deposition of the hematite-chamosite ore, the sea was restricted by some means to a basin like form, but was still sufficiently shallow to allow the migration of a fauna found in similar rocks in Europe. During the deposition of the material in which the oolitic pyrite is found in Zone 3, a sufficient depth must have been attained for open currents to pass, carrying marine plankton as is shown by the graptolites which occur in the oolitic pyrite rock.

7. *Evidence from fossil algae in ore on its primary origin.*— The role which minute organisms such as algae and bacteria play in the building up of such deposits, has not as yet been very thoroughly investigated and it is probable that many of the little known chemical processes may be due to the action of such organisms.

"According to Ehrenberg, the algae especially the so-called iron algae, *Gallionella ferruginea*, Ehrenb., are active ore precipitants coating their cell walls with ferric hydrate and opaline silica. These algae are abundant on the sea bottoms. According to the recent works of Mollisch and Winogradsky, these and most other supposed algae are ciliated bacteria of different kinds, especially *Leptothrix ochracea*." Beck (2: p. 102).

Tubules of algae are found abundantly in the fragments of brachiopod shells in the ore, some of which extend out of the shells into the surrounding matrix, in such manner as to indicate that they pierced the material of the shells after the latter were deposited as fragments in the soft sea bottom. The algae are also found plentifully preserved in ferruginous phosphatic nodules and many of them have been found in spherules of hematite and chamosite.

Most suggestive of all, however, in relation to the diagenesis of the ore, is their presence in siderite, in which, in the upper part of the Scotia bed, tubules of algae have been preserved.

These fossil algae have been studied by Prof. Gilbert van Ingen who refers them to the Schizophyceae or Blue Green Algae. Dr. Marshall A. Howe, of the New York Botanical Garden, has examined the algae and remarked (9) upon their close resemblance to species of modern blue green algae of similar habits. When living, such algae take up carbon dioxide and water which are broken up by the action of light (photosynthesis), and oxygen given off as waste products. (John M. Coulter, 6: Pt. I, p. 150; Pt. III, p. 302).

The chemical reaction may be outlines as follows:

$$12\ CO_2 + 12\ H_2O = \underset{\text{Starch}}{2\ C_6H_{10}O_5} + \underset{\text{Oxygen}}{12\ O_2} + 2\ H_2O$$

The carbohydrate, starch, thus obtained by photosynthesis, and nitrogen obtained by the process of osmosis, are converted into proteids which provide food for the organism and build up its protoplasm. (H. W. Conn., 5: p. 129).

The food of the plants consists of CO_2, H_2O, nitrates, phosphates, potash salts and other minerals in small quantities, (Conn, op. cit). During the life of the algae oxygen is their most abundant waste product, and a zone of oxydation would be found in the immediate vicinity of the flourishing plants. The decomposition products of the algae, together with those of the brachiopods, trilobites, worms and other animals present in this Ordovician sea, provided a large amount of carbon dioxide and ammonia together with small amounts of other gases and salts.

The tubules of the fossil algae are frequently covered exteriorly with a layer of crystalline hematite. This is especially noticeable where spherules of chamosite have been bored and it seems probable that the oxidation of the iron was in these instances at least, caused by the life processes of these algae, and since by examination of only a small number of slides, the algae have been found at many horizons in both the Dominion and Scotia beds, and evidently lived in great abundance throughout the time of deposition of the ore strata, it may be that sufficient oxygen was provided in this manner to account for the large proportion of the iron of the ore in the form of hematite.

The iron, brought into the sea by streams, in the form of salts of organic acids, chlorides, sulphates, ferrous bicarbonate, etc., having a higher specific gravity than sea water, would sink to the bottom, and ammonia formed on the sea bottom from decaying organic matter, might produce directly or indirectly, a precipitate of iron hydroxide.

The oxidizing action of the algae above referred to would tend to oxidize the precipitated hydrate of iron to hematite. Since the oolitic ore occurs with argillaceous rocks, hydrated silicates of alumina were undoubtedly abundant and the chamosite and other silicates of iron, may have had their origin by the combination of the iron and hydrated aluminous silicates, the process taking place contemporaneously with the formation of hematite.

Prof. Lindgren (10: p. 250) cites an occurrence on the south side of Molokai, Hawaiian Islands, where hematite mud is spread out over a large area of shallow coral reef.

Ehrenberg has found that certain unicellular plants coat their cell walls with ferric hydrate and opaline silica. This phenomenon suggests the possibility of a common source in the plant activities for the silicates as well as the oxide of iron. As I have found no evidence of organic influence in the building up of the concentric layers of hematite and chamosite, I think that this structure is probably due to physical causes.

9. *The Formation of Oolites by Physical Causes.*—Dr. T. Wayland Vaughan (13: p. 303) has studied the processes which result in the formation of oolites composed of calcium carbonates which are forming at the present time in shoal waters off the coasts of Florida and the Bahama islands and has announced the following conclusions:

"The empirical facts in the process of the formation of the Floridian and Bahaman oolites are demonstrated. They are as follows: (1) Denitrifying bacteria are very active in the shoal waters of both regions and are precipitating enormous quantities of calcium carbonate which is largely aragonite; (2) this chemically precipitated calcium carbonate may form spherulites which by accretion may become oolite grains of the usual size, or it may accumulate around a variety of nuclei to build such grains."

Similar physical methods were probably operative for the formation of the oolitic hematite-chamosite. The tendency of extremely minute particles to collect together under the influence of surface tension, is well known and the micro-crystalline hematite and chamosite have together furnished such pulverulent material for the production of the spherules in this manner.

10. *Origin of Siderite and its Relations to Hematite and Chamosite.*—While oxidizing conditions obtained on the sea bottom, very different conditions were produced in the underlying sediments. Wherever, in the Wabana ores, siderite has been found in association with hematite and chamosite, it replaces them, but the tubules of the algae are frequently preserved in it intact. As has been shown above, it is probable that oxidizing conditions obtained upon and immediately above the sea bottom, due to the growth of algae. At the same time some of the dead and decaying algae together with inarticulate brachiopods, worms, trilobites and other organisms would be entombed by the upper portions of the newly formed sediments. During the deposition of the ferruginous beds, it seems probable that a variety of iron salts were in solution in the sea water, and ammonium carbonate resulting from the decomposition of organic matter, where entombed by overlying sediments, may have decomposed these iron salts, with the formation of the corresponding ammonium salts and the precipitation of the iron carbonate, siderite.

Quartz grains and chamosite, both in matrix and spherules, are frequently partly destroyed and replaced by siderite. Solutions of alkali carbonates and hydroxides have a considerable solvent action on quartz and dissolve non-crystalline silica very readily, and such a solvent may have been produced and concentrated locally in sufficient quantity to aid in the solution of these constituents, the iron carbonate being deposited. Thus it appears that while the hematite and chamosite were being deposited on the surface of the bottom deposits, siderite was contemporaneously formed in the underlying hematite-chamosite muds, for a distance downward, the amount of siderite formed being dependent upon the quantity of imprisoned decomposition products.

Origin of the Pyrite Beds

At the close of the deposition of the Dominion Bed at the top of zone 2, a disconformity occurs immediately above which, layers of oolitic pyrite alternating with beds of fissile black shale are found which hold an abundant fauna of graptolites, (identified by Dr. Rudolph Ruedemann in a personal communication to the writer in December, 1912 to be *Didymograptus cf. nitidus*,) associated with brachiopods, orthoceratites and probably other fossils.

Since the graptolites were planktonic organisms drifted about by ocean currents, their presence indicates a sudden deepening of the sea at the close of the formation of zone 2. No hematite accompanies the pyrite and an entire change takes place from sediments containing oolitic hematite and chamosite with siderite, to oolitic pyrite, a few inches of shale intervening. With the deepening of the sea, the algae would cease to grow, for they are shallow water plants depending on sunlight for their life processes. The oxidation produced by these organisms would therefore be lacking and conditions similar to those found on modern sea bottoms where pyrite is forming, were probably produced.

The formation of (FeS) ferrous sulphide in the bottom deposits of the Black sea is described by E. Androussow (1: p.7). He has found that a micro-organism, *Bacterium hydrosulfuricum ponticum*, liberates hydrogen sulphide not only from albuminoids but also directly from sulphates and sulphides, and ferrous sulphide results from the action of the hydrogen sulphide on iron salts. Other micro-organisms as yet insufficiently studied, also occur in the Black sea. In depths of 300 to 717 fathoms two varieties of mud were found, a viscos sticky black mud holding iron sulphide (FeS) and a less dense blue mud holding pelagic diatoms, a smaller quantity of FeS and concretions of FeS_2. The blue mud is thought to underlie the black mud. Thus the fact is established that FeS_2 is produced in bottom deposits of the Black sea at depths of 300 to 717 fathoms.

CONCLUSIONS

The oolitic iron ore with ferruginous shales and sandstones forms part of a series of sedimentary rocks of Lower Ordovician age.

The ore beds are characterized by ripple marked surfaces and cross-bedded layers, and contain remains of animals which lived in shallow water. The spherules of the ore vary in size from .1 to .5 millimeters and are composed of alternating concentric layers of hematite and chamosite. These spherules were pierced by living boring algae, hence the iron minerals forming them were precipitated near the surface of deposition, while the algae flourished on the sea bottom.

Siderite occurs in smaller quantity than hematite and chamosite, but becomes locally abundant. It replaces the hematite and detrital quartz in the ore. The algae are found in all horizons in the ore beds and formed a very abundant marine plant life growing on the sea bottom. Tubules of the algae are preserved in the siderite and are frequently coated exteriorly with hematite. The siderite was chemically precipitated, probably under cover of overlying sediments where concentrations of ammonia and carbon-dioxide resulted from decaying organic matter. Thus while hematite and chamosite were forming at the surface of deposition, the siderite was contemporaneously formed in the immediately underlying sediments.

There is a total lack of limestone from the series, and igneous rocks are also absent. Practically all of the original calcium content of the ore, averaging about 2.5 per cent is present in the form of fossil remains composed largely of calcium phosphate, or as calcium phosphate derived from such organic matter. The phosphorus of the ore is also derived from the remains of organic life preserved in it. No evidence of diagenetic transformation from an original oolitic limestone to an oolitic iron ore has been found and no concentration of iron has occurred since the deposition of these ferruginous sediments. They are primary bedded iron ore deposits, mined today in essentially the same condition except for induration, faulting, and the addition of

small amounts of secondary calcite and quartz in fault cracks, as when they were laid down.

Oolitic pyrite also occurs as part of the same series of sediments, but is characterized by a planktonic fauna indicative of open ocean currents and deeper water. The layers of pyrite show distinct stratification and are probably similar in origin to modern deposits of pyrite forming in the Black Sea.

The pyrite spherules are composed of concentric layers of pyrite, frequently alternating with layers of phosphatic material. Pyritized and unpyritized graphtolite and brachiopod remains occur together in contact with the spherules, indicating that some mechanical mixing took place on the surface of deposition.

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NOTES ON THE GEOLOGY OF THE "MOLLY" MOLYBDENITE MINE, LOST CREEK, NELSON MINING DIVISION, B.C.*

By Chas. W. Drysdale.

Western Branch Rossland Meeting, 1915

The war has created a great demand for molybdenite ores; for, besides its use in the preparation of molybdates used as a laboratory test for phosphoric acid and as a dye for certain kinds of leather, rubber, silk and wools as well as in the manufacture of a blue pigment in the porcelain industry, molybdenum is extensively used in the steel industry for hardening armour plate, in the manufacture of rapid firing guns, and for all purposes requiring a high speed steel. It is also used as a stabilizer for high explosives, to prevent their deterioration and premature explosion.

Mineralogy.—The mineral molybdenite (MoS_2), a sulphide of molybdenum containing 60% of molybdenum, is a foliated grey to light blue mineral with similar coloured streak; it belongs to the hexagonal system of crystallization; and has a hardness of 1.3 and specific gravity of 4.6 (biotite mica is 2.9 and graphite 2). Among the minerals, molybdenite resembles graphite most nearly, but can be readily distinguished from graphite by its less massive character, its streak which is grey to light blue in colour instead of black as in the case of graphite, and its weight which is twice that of graphite.

The mineral wulfenite ($Pb\ MoO_4$), a molybdate of lead containing 26.2% molybdenum, is a massive, resinous light grey to yellow mineral belonging to tetragonal system of crystallization, and having a hardness of 3 and specific gravity of 6.8.

*By permission of the Director of the Geological Survey of Canada.

Market Conditions.—The normal price of molybdenite before the war was from $8 to $10 a unit for an 85% MoS_2 ore, which, at $9 a unit is 38¼ cents per pound of the ore, or 45 cents per pound of molybdenite contained in the ore. At the time of writing (July 13th, 1915) the molybdenite ores are graded as follows:—

20% molybdenite $10 per unit of MoS_2 per ton of 2,000 lbs., 10c. per lb. of ore, which is equivalent to 50c. a lb. for the pure molybdenite calculated to be contained in such ore.

40% molybdenite $12 per unit of MoS_2 per ton of 2,000 lbs., 24c. per lb. of ore, (60c. lb. for pure molybdenite).

65% molybdenite $15 per unit of MoS_2 per ton of 2,000 lbs., 48¾c. per lb. ore, (75c. lb. for pure molybdenite).

95% molybdenite $20 per unit of MoS_2 per ton of 2,000 lbs., 95c. per lb. at ore, ($1.00 lb. for pure molybdenite).

Molybdenite sold as high as $1.50 per pound last February and sales have been made at $1.10 per pound. It has been rumored that it sold in London as high as $1.75 per pound or $3500 per ton.

On June 5th and July 3rd the market quotations for a (95% MoS_2) ore were a minimum of $1000 and a maximum of $1500. The market now exacts a penalty of 50c. per unit for each unit of copper in excess of 1%. The same penalty is exacted for tungsten. Thus a 40% ore if containing 1% of copper only, would sell at $12.00 per unit; whereas ore having the same molybdenite content, but containing 7% of copper, would bring only $9.00 per unit. Arsenic, bismuth and other volatile elements are not particularly deleterious and no penalties are exacted for them.

The world's production of molybdenite for the years 1902 to 1913 is shown in the following table:

Production of Molybdenite*

(In Metric Tons).

Compiled from Mineral Industry, 1913.

	New South Wales	Queensland	Norway
1902	16	c42	20
1903	31	c24	31
1904	26	c22	30
1905	20	64	46
1906	34	108	1026
1907	22	68	30
1908	9	89	35
1909	29	94	30
1910	50	108	
1911	21	c232	
1912	57	104	21
1913	80	68	

c. Includes bismuth and some tungsten.

In a contribution to "Mineral Resources of the United States, 1913, Part 1, p. 241," Frank L. Hess remarks: "No commercial production of molybdenum is known to have been made in 1913, and no promising new deposits were reported during the year. The demand, however, suddenly increased, and in the latter part of the fall a French firm offered the following prices for molybdenite:

Percentage of MoS_2	Price per unit in long tons
50	$ 5.00
60	6.00
70	7.00
80	8.50
90	10.00
95	12.00

*Compiled from Mineral Industry, 1913.

"Still higher prices were quoted in some of the technical journals. In Queensland £326 ($1586) a long ton was paid for high-grade molybdenite in December, 1913. (Min. Eng. Rev. (Melbourne), Vol. 6, p. 170, 1914). Earlier in the year Australian molybdenite of high grade had sold in this country for 45 cents a pound. Some ore was brought to this country from British Columbia for concentration and was shipped abroad. No record is kept of the molybdenum ores imported. Both metallic molybdenum (powder) and ferromolybdenum are made in this country, besides the very refined ductile metal made by the General Electric Co. The metallic powder sold for about $1.50 a pound. Molybdenum and ferromolybdenum, valued at more than $200 a ton, were imported during 1913 to the extent of 7.8 short tons, valued at $15,939, against 3.9 tons, valued at $4,670, in 1912."

Canadian Occurrences.

Molybdenite occurs at a great number of localities in Canada but it appears to occur in commercial quantities in only a few places. Mr. R. A. A. Johnston, Mineralogist of the Geological Survey, has kindly supplied a list of sixty different localities in Canada where molybdenite occurs, and of these twenty are in British Columbia, eighteen in Ontario, fifteen in Quebec, three in Nova Scotia, three in New Brunswick and one in Manitoba. The British Columbia localities are as follows:—

Nelson Mining Division.—Molly Mine, Lost Creek; Bear Creek (a branch of Sheep Creek), and Granite, four miles west of Nelson.

Trail Creek Mining Division.—A massive granular form has been noted at the Giant and other Rossland mines, and in the deep levels of the War Eagle mine where it is intimately associated with free gold. It is also reported from Big Sheep Creek.

The Boundary District.—In association with copper ores in the district.

Grand Forks Mining Division.—Franklin Camp, disseminated in the upper border of the granodiorite mass.

Similkameen Mining Division.—Bate's Camp, Aspen Grove and the Nickel Plate mine.

Kamloops Mining Division.—Highland Valley, associated with copper, and at a point three miles south of Grande Prairie.

Osoyoos Mining Division.—Keremeos.

Lillooet Mining Division.—Lillooet and Bridge Rivers and in range southwest of Seton and Anderson Lakes.

Yale Mining Division.—Divide between Spuzzum Creek and Harrison Lake.

New Westminster Mining Division.—Salmon Arm, associated with copper ores at Stave Lake about 22 miles above the power dam, and seven miles above the head of navigation.

Victoria Mining Division.—North Side of Mount Buttle, Vancouver Island.

Nanaimo Mining Division.—Carrington Bay, Cortez Islands, Strait of Georgia, in small quantities in quartz veins; Malaspina in a copper bearing vein, and Texada Island.

The "Molly" Mine, Lost Creek.

The Molly Mine is situated on Lost Creek, on the old Dewdney Trail, about 15 miles by wagon road from Salmo. It is at present being opened by a Vancouver syndicate. The property includes a group of four Crown Granted claims—"Molybdenum No. 1," "Molybdenite," "Bromyrite" and "Bromyrite King." The claims were located July 15th, 1913, by S. N. Ross, H. E. Bennett and J. A. Benson.

Production.—Since discovery, 50 tons of molybdenite ore have been shipped from the property. The first car of 24 tons was shipped by the owners to Denver, Colorado, on Oct. 1st, 1914. The average run of the shipment was 16.586% molybdenite and 822 pounds of the ore averaged as high as 30.175 per cent MoS_2. Specimens of ore of this grade assayed $2\frac{1}{2}$ oz. in silver with a trace of copper and gold. Selected samples assayed from 52% to 80% MoS_2. At that time the shippers were paid for 85% of the MoS_2 contents at the low rate of 20c. per pound

On March 10th, 1914, the property was leased to G. H. and J. P. Bell of Salmo, who proceeded to actively develop it; and on April 9th, 1915, shipped one car of 24 tons to Denver, Colorado. The shipment ran 12.26% MoS_2, and 90% of the MoS_2 content was paid for at the rate of 50c. per pound. In addition 2 tons of ore samples, which assayed 9.5% MoS_2 and sold for $1.00 per pound of MoS_2 content, were shipped by the owners to New York. Several thousand tons of milling ore of about probably 4% MoS_2 lie on the mine dumps. During the spring of 1915 the property was bonded for $100,000 by a Vancouver syndicate, which continued the development of the mine, intending, should conditions warrant, to presently install a small concentrating plant on the property.

Geology of Ore Deposit.—Like so many Cordilleran ore deposits, that of the Molly mine is associated with the upper border of a large intrusive mass or batholith of granitic rocks which have been laid bare by erosion. The deposit may be classified according to form as stock works and impregnations of molybdenite, pyrite and pyrrhotite, about the upper borders of the granite stock; or as a cupola stock—a comprehensive term used by Daly to indicate a connexion in depth with a main underlying batholithic mass. This cupola stock is composed of a quartzose granite of post-Jurassic age, almost without mica. The granite is cut by pegmatite dykes or veins which appear to lie in a direction parallel to that of the ore zone. The ore at the surface weathers to limonite, kaolin pink hydrous molybdate of iron which stains the decomposed granite along joint planes and furnishes the prospector with a good indicator of molybdenite ore. The chilled border (endothermal effect of granite intrusion) of the stock forms a hard shell or 'capping' averaging about 6 feet in thickness and composed of a fine grained aplitic variety of granite dotted through with a few scattered molybdenite grains. The thickness of this capping, however, varies a great deal from place to place and in certain localities it has been entirely removed by erosion. The variability of the thickness is due both to the irregularity of the borders of the stock and to thermal conditions at the time of intrusion. In the bottom of the main open cut the capping is about ten feet thick and steeply dipping with the border of the stock, whereas higher up the hill

it lies with gentler dips and varies from three to six feet in thickness. The granite is intrusive into the Pend d'Oreille schists and limestones which Daly refers to the Carboniferous period. The Pend d'Oreille roof or cover formation at the west border of the cupola stock is composed of dark argillaceous schists and at the south-east border of crystalline limestone, the whole series being severely metamorphosed in contact with the granite intrusive. This roof formation strikes in a north-westerly and south-easterly direction and has steep dips to the south-west. Associated with the limestone is the so-called "Iron Dyke" which first attracted the attention of prospectors who many years prior to the location of the Molly Group had noticed flakes of molybdenite in the roof rocks of this vicinity. The "Iron Dyke" is a replacement deposit of pyrite and pyrrhotite in a lime-silicate gangue, having small inter-areas of crystalline limestone. In this contact-metamorphic zone (exothermal effect of granite intrusion) occur minute flakes of molybdenite but not in commercial quantities. The Pend d'Oreille formation in places may be only a thin covering overlying the ore-bearing granite stock. The molybdenite ore-zone lies beneath the hard shell of fine grained granite or 'capping' and occupies that portion of

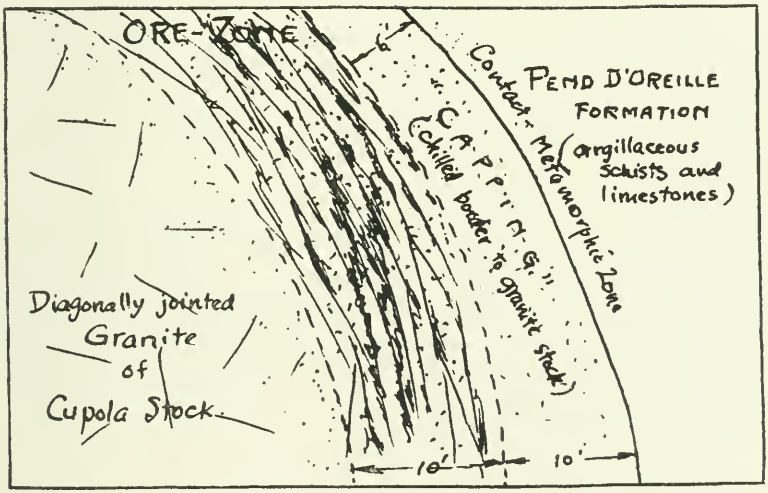

Fig. 1.—Shows occurrence of ore in ore zone. Black-molybdenite ore. Diagrammatic.

the cupola stock characterized by platy or sheeted jointing[1] in which the joint planes are closely spaced, interfinger, and lie in a general parallel position to the contact border (Fig. 1). The joint planes in the ore-zone dip at low angles away from the centre of the stock-dome, which lies up the hill eastward and above the present workings and in part covered by the Pend d'Oreille formation. The molybdenite ore lies along these interlocking joint planes in reticulating veinlets branching and crossing from joint to joint as well as impregnating the granite between the planes of jointing. The granite in the ore zone in places becomes blocky and contains less orthoclase (potash) feldspar than the sheeted granite. When such is the case the ore becomes leaner (milling ore) and even barren of value.

The ore zone appears to be about 10 feet thick as exposed in the main open cut[2] at the west border of the cupola stock. The granite below the ore zone becomes more massive, blocky and coarser in grain. Whether the molybdenite ore persists into the diagonally jointed granite or is only confined to the platy jointing still remains to be seen. The granite, however, for considerable distances from the ore zone is impregnated with molybdenite and much of it might be milled profitably.

The ore zone was first discovered near the centre and to the north of the stock-dome, where, through the truncation of a spur by the Lost Creek valley glacier the capping of hard granite shell and thin veneer of softer Pend d'Oreille rocks had been stripped and the ore-zone laid bare for the prospector's stake. Most of the development work has been carried on at the western border of the cupola stock where the capping had to be stripped and the ore-zone opened by means of a quarry. The ore-zone has been traced and in part proved at the surface for about 1200 feet along the side of the hill and the manner in which the closely spaced, massive, joint-planes dip flatly with the hill some 800 feet east of the main workings suggests that the centre of the stock-dome is further up the hill and probably capped by the Pend d'Oreille formation.

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[1]This sheeted or plated jointing gives to the granite an almost sedimentary appearance.

[2]This open cut is about 75 feet deep and 10 feet wide at the bottom where the working is still (July, 1915) in high grade ore (specimen) (Plate II).

Origin and Classification of Deposit.—From the above described field relations it may be inferred that the molybdenite is slightly younger than the enclosing granite, although from the same parent source, and either accompanied or followed the intrusion of the pegmatite dykes which represents the latest crystallization of the granite magma. In this pegmatitic magma, molybdenum disulphide was probably a prominent constituent, in fact some granites have molybdenite as an accessory mineral and primary. It is thought that the molybdenum was in gaseous as well as liquid solution and thus had sufficient penetrating power to impregnate the dense granite as it has apparently done for considerable distances from the actual veinlets. The molybdenite appears to be the result of impregnation at time of or following pegmatitic intrusions rather than an original constituent of the granite. If this be the case the heaviest deposition of the mineral might reasonably be expected where the granite is most cut up by pegmatite dykes. According to origin then this deposit would be classified as belonging to the pneumatolytic type of epigenetic ore-deposit similar to the geological occurrence of certain ores of wolfram, tin, tungsten and other metals.

Future Work.—The most economical way to recover the ore and to develop the property at present would appear to be by removing the 'capping' and mining by open cut and quarry methods. On account of the flatly dipping character of the ore body above, its rudely circular form and uncertain nature, underground, development by means of cross-cuts and drifts would prove expensive and precarious at the present stage of development.

The geological structure, the small amount of mica in the ore which contains also only traces of copper, and the existence of a sufficiently ample water supply are factors favourable to the provision of a mill to treat the lower grades of molybdenite ore. By thus raising the percentage of molybdenite through concentration to say 95% MoS_2 not only the highest prices would be obtainable but also a more widespread market and sale for the product made possible.

THE ROCKS OF THE PORCUPINE DISTRICT

By Alfred R. Whitman.

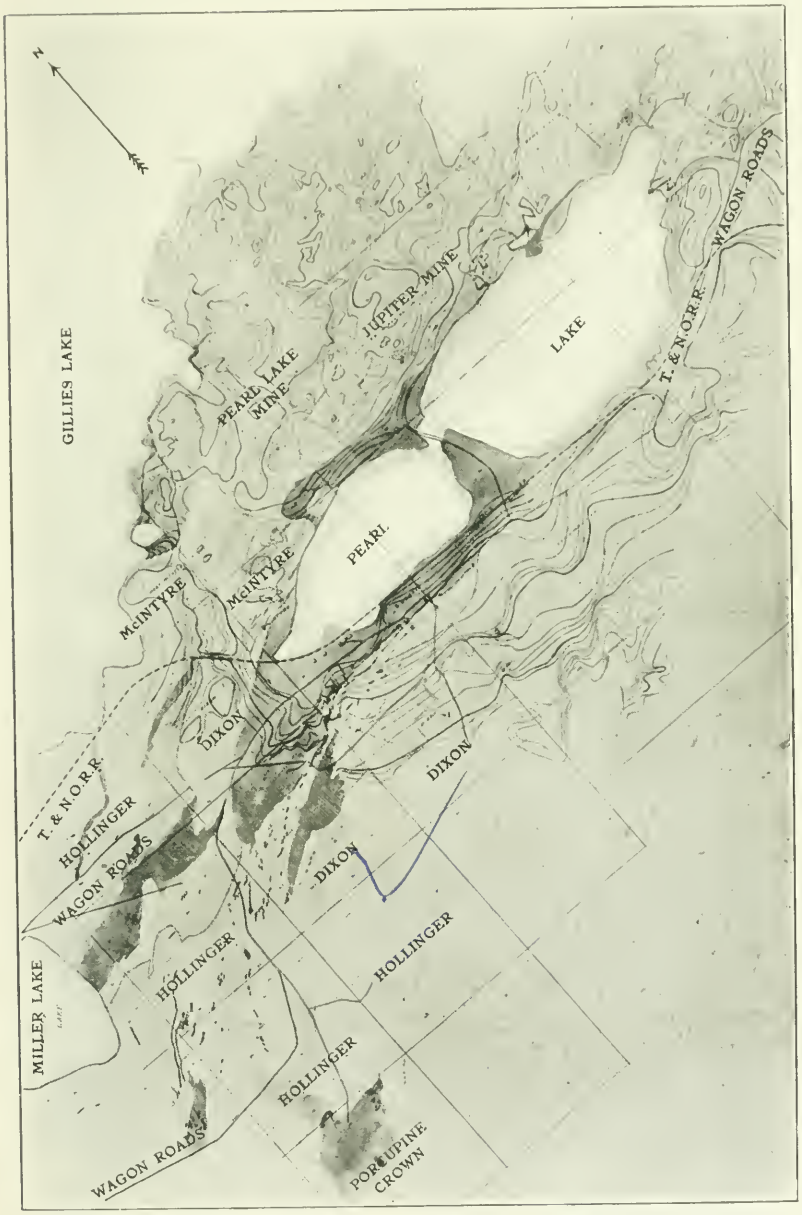
The present paper in intended to be in no sense exhaustive; and is submitted with apologies for its incompleteness. The writer has in fact, confined himself to describing some of the rock phenomena occurring especially in the Pearl Lake area and its neighbourhood.

There are many problems in connexion with the geology of the Porcupine district that have not as yet been clarified. This is due mainly to the extreme alteration of the rocks of the district.

There are two methods of attacking these problems; (1) by microscope, and (2) by close field study. Having employed both methods on the surface and underground and having studied the reports of previous investigators the writer has arrived at the conclusions hereunder presented.

Ellipsoidal Basalt.—The most widely distributed and seemingly, the oldest rock of the Keewatin igneous series is a chloritic schist, known as 'amygdaloidal basalt' or 'ellipsoidal greenstone,'[1] and commonly regarded as representing submarine flows. Within the area covered by the present studies this rock has everywhere been found to be too much altered to allow of definite classification from miscroscopic examinations, since no original minerals are recognized except perhaps a few laths of plagioclase and some titaniferous magnetite. But on account of the ellipsoidal structure, fineness of grain, and the presence of amygdules it is assumed to be a basic lava. Its wide distribution and sheet like form also suggest its effusive character. However, the ellipsoidal structure is not equally well developed in all parts of the field. In fact, though found here and there over a very much larger area than that covered by this entire district, the ellipsoidal structure in its most pronounced development is limited to certain areas; an equal amount of area, perhaps, being

[1] "The Porcupine Gold Area" Ont. Bur. Mines, 21st Ann. Rep. Vol. 21, Pt. 1.



Photograph of Map of Pearl Lake area (courtesy of McIntyre Porcupine Mines, Limited),

Dark shaded areas = Quartz Porphyry
Light " " = Greenstone and Gray Schist
Unshaded " = Ellipsoidal Basalt

covered by rocks of similar origin and general appearance but lacking this peculiar structure.

Ellipsoidal lavas are generally basaltic; and it is known that where basaltic lavas flow into the sea or are erupted beneath it, as at Hawaii, an ellipsoidal structure is developed. However, at least at one locality within the writer's experience dykes of ellipsoidal basalt have been recognized. That locality is San Francisco Bay, California, where the Franciscan radiolarian cherts are intruded by dykes of this kind on the Marin and San Francisco Peninsulas, the dykes containing included blocks of chert exactly similar in appearance to that forming the walls.

That structure, as has already been remarked, is not equally well developed everywhere in this field, and where not attaining its best development shades off gradually through poorer and poorer developments to complete obscurity. If it were due to submarine eruption this would be a difficult condition to explain. On the Hayden property a few miles south west of the Pearl Lake area, beds of rock of this series were found, which had a fragmental structure and are believed to represent ancient tuffs. Also it has been recognized at the Porcupine Crown, the Hollinger, and the Dome mines, that the microscopic character of all the supposed 'basalt' is not the same, being more acid at some points that at others.

If the norm of this rock is taken as an andesite, of subaerial effusion, all these seemingly contradictory phenomena are more readily harmonized, for a series of eruptions from an andesitic magma usually gives rise to flows of varied acidity alternated with pyroclastic eruptions. Then if the ellipsoidal structure is interpreted as being due in some way to the basic character of the lava, and not to the environment encountered upon eruption, the gradations of its development may be understood as due to the gradations in basicity of various flows, and it is not necessary to invoke the agency of the sea in which tuffs must undergo a size classification like other sediments, as this is not the case in the instance mentioned.

The Greenstone.—For want of a better name, this term is here used to designate a chloritic rock of less foliated character than the basalt and always lacking in any semblance of ellip-

soidal structure. Whereas the basalt is extremely schistose in texture, this rock is relatively massive. Both may be classed as fine-grained rocks, but the greenstone is usually distinctly granular, weathering with a sandy surface.

It can be distinguished from the non-ellipsoidal rocks of the basaltic series not only by its more massive and granular texture, but also by the fact that it also shows distinct contacts with the latter. These cut across the ellipsoids in the more basic variety indiscriminately, and appear to be of instrusive origin. The idea of intrusive origin is also borne out by the elongated form of the areas, in plan. Over the limited field represented by the accompanying map, the greenstone areas are distinctly elongated in the direction of foliation of the basalts. This has been observed, though not demonstrated by mapping, over a much larger area, and in fact wherever coming under the writer's observation.

Under the microscope the greenstone shows more character than the basalt, usually containing small areas of hornblende, larger amounts of titaniferous magnetite, and considerable amounts of leucoxene. Now and then one finds a few shadowy laths of plagioclase. The other minerals are the same as those found in the basalt, chlorite usually being slightly more conspicuous. They are sericite in small amounts, secondary quartz, and frequently considerable quantities of ferro-dolomite and pyrite. Under the microscope there is as great a contrast in foliation as is observed in hand specimens.

Carbonate Schist.—Under this name and also that of 'grey schist' are classed greyish schists of fine grain which contain over fifty per cent of ferro-dolomite. These schists appear to have been chloritic originally, and to have acquired their present character by the diffusion through them of ferro-dolomite. They usually betray themselves in the field by the brownish tinge developed by the breaking down of the carbonate and the oxidation of the iron to limonite through the action of the atmosphere.

This carbonate schist is usually found in its best development near intrusive masses of quartz-porphyry schist. Frequently it shows quartz eyes which look in the hand specimen like

phenocrysts, and might lead one to regard it as a dacite, since basicity is indicated by small quantities of leucoxene and chlorite. However all the other characters of this rock are similar to those of the basalt and the greenstone.

The quartz eyes have been found to be most abundant near contacts with the quartz-porphyry, and in many thin sections most of them have proved to be of secondary origin. Although frequently under the microscope quartz areas can be pointed out which can not be proven secondary, and which strongly resemble original phenocrysts; nevertheless the writer believes that all are of secondary origin, and the rock merely a carbonated phase of the other two rocks described as basalt and greenstone, the carbonation having succeeded and resulted from the intrusion of the quartz porphyry, since it is areally associated with this rock.

Quartz Porphyry.—This term was applied in the first reports[2] to a light coloured coarse grained rock very conspicuous in the mineralized areas. It is a porphyry of medium grain, the larger phenocrysts of feldspar frequently attaining a length of three-eighths of an inch, the ground mass now being obliterated by alteration.

Generally the rock is highly foliated, but at some points it is quite massive. Where least altered, it is seen to have quartz phenocrysts ranging in size up to $\frac{1}{8}$ inch in diameter, and feldspar phenocrysts as stated, consisting chiefly of albite or an acid plagioclase near it in composition. Ferro-magnesian constituents are lacking, although now and then a few small grains of leucoxene seem to indicate the former presence of a titaniferous mineral. The proportions of the secondary minerals, sericite, ferro-dolomite and quartz, vary considerably, from place to place, at some points the first mineral being so abundant as to make virtually a sericite schist. At other points the rock is seen to be highly carbonated. Pyrite also is abundant near the veins and frequently elsewhere, in the form of small cubes.

The rock is intrusive into the others, taking the form of stocks. The shape of these masses as cross-sectioned by the surface is roughly elliptical, the long axes trending a few degrees

[2]Ont. Bur. Mines. 21st Ann. Rep., Vol. 21, Pt. 1.

more north of east than the strike of the schist foliae. Not enough work has been done underground, to fully expose the forms of the stocks, but it has been shown that they are not of smooth outline, nor even of compact form. In plan they send

Bewick Moreing property; École showing transition into feldspar grains diffused through greenstone; cross section.

off apophyses into the walls, in certain measured cases being forty or fifty feet long and twenty-five feet through at the base. One of these is penetrated by the main drift on the 300 foot level of the McIntyre mine in the E block. Again masses have been

found to apex beneath the surface and also to bottom within the mine workings. Cases of this have been noted by Mr. W. H. Wiley in the Hollinger mine. Also on the McIntyre property in the K block north of Pearl Lake, a crosscut driven southward on the 600 foot level found the north margin of the Pearl Lake stock 250 feet south of its outcrop on the surface, showing the mass to flare outward near the surface, like the under side of an eroded laccolith. Other workings of the McIntyre mine, and also of the Porcupine Crown, have shown the margins of the stocks to be deeply embayed and studded with projections, making them extremely irregular; and there is a general tendency of the offshoots to conform roughly to the foliation of the older rocks.

All these peculiarities of form as well as the fact that near the southeast shore of Gillies Lake a large inclusion of schistose ellipsoidal basalt is seen in the quartz porphyry are taken as indicating that the older rocks were in a schistose condition at the time the quartz-porphyry was intruded. This conclusion is further borne out by a peculiar marginal injection phenomenon somewhat resembling lit-par-lit, which not being described from other localities the writer ventures to name *école injection*, from the fact of its consisting of swarms of little cigar-shaped pipes apparently distinct from one another, and all uniformly oriented, like a school of fish in transit.

École Injection.—The French word for school or swarm was chosen to designate this form of injection instead of using the English word, merely for correspondence with the French expression lit-par-lit which names a phenomenon very similar in character and long since recognized as a type.

The members of an *école*, as stated, are distinct cigar-shaped or fish-shaped masses uniformly oriented among the foliae of the schist which they penetrate, their long axes coinciding with the pitch of the flutings of the schist. These masses or pipes as they may be called, range in size from eighteen inches in diameter down to minute masses half-inch and less in size. The length varies in proportion to the cross-section from two to ten or more times its measure.

The best known example of the phenomenon is found on the Porcupine Crown property northeast of the main shaft, and north-

west of the office, extending over the property line onto the Hollinger. In the former case the quartz porphyry mass on whose northeast margin the phenomenon occurs, is full of inclusions of wall rock, in the form of slabs, many of them the

West Dome; typical école in cross section conforming to foliation of schist

size of a man's hand, oriented parallel to the foliae of the undisturbed schist, and to the pipes of the adjacent *école*. The contact of this mass with the adjacent basaltic schist is ill-defined, there being a gradual transition from the quartz porphyry filled

Porcupine Crown; showing quartz-porphyry rich in inclusions

with intrusions to the *école*, in which the matrix is green schist, and the white quartz porphyry masses are in the minority. In some places the tongue-shaped *école* has a well defined contact with the undisturbed basaltic schist, while in other places the pipes grow smaller until they dwindle to single crystals of feldspar, and merge through a zone of transfusion into schist not visibly affected.

West Dome; breccia

In the second area mentioned, the transitions are not so clearly displayed, although each stage is better developed, and the *école* is much larger, covering an area approximately 100 feet by 75 feet.

Three or four other such areas are found on the Hollinger and Dixon properties on the margins of quartz porphyry stocks. Other smaller ones are found on the eastern extremity of the

Gillies Lake stock near one of the houses belonging to the Pearl Lake mine, in the hollow north east of the Pearl Lake shaft, and between that shaft and the house mentioned. These last two occurrences are unassociated with any visible outcrop of quartz

West Dome; sharply defined zone of breccia and école members: the knife shows direction of foliation of schist

porphyry, although they are in north-easterly extension of the southern portion of the Gillies Lake stock. Another group of them is found at the east end of Pearl Lake on the knob occupied by the Bewick-Moreing bungalow, being immediately north and

east of it. A very large typical *école* is to be seen on the West Dome, and the Dome property south-west of the mill, in the greenstone area north of the quartz porphyry. It is one hundred feet or more wide and more than a thousand feet long, striking with the schists a little north of east. Here all the phenonena are exhibited, with the exception of the transition into the inclusion-rich margin of the quartz porphyry.

Bewick-Moreing property; The lighter coloured ores represent poorly developed école members in breccia

Other smaller areas have been recognized on the hill crossing the Dome Extension, on and near the contact of the quartz porphyry with the greenstone.

All these occurrences, and others not mentioned because of the difficulty in some cases of distinction from conglomerate or pyroclastic rocks where weather and glaciation have combined to

make obscurity, can best be collectively described under the ideal case which follows.

From an area of quartz porphyry a tongue extends outward into the sorrounding greenstone parallel with the foliation. On

West Dome property; Weathered breccia

its margins and over the end, inclusions of wall rock are abundant, and particularly so toward the indefinite contact. The inclusions are slabs, usually of lenticular cross-section, though frequently tabular with angular ends. They vary from the size of an almond

to that of an army canteen, or larger, and are uniformly oriented, parallel with neither side of the tongue, but with its axis, which is that of foliation. Toward the margin they become more abundant until quartz porphyry matrix gives way to greenstone

West Dome; Isolated large école member amid smaller ones; cross section; 25c piece shows scale

matrix with pipes of quartz porphyry thickly scattered through it, the major and intermediate axes of the pipes having the same orientations as those of the inclusions in the acid matrix. Finally at a distance of from five to fifty feet from the approximate con-

tact, the white pipes become fewer and disappear, sometimes at a definite contact and sometimes not.

Beyond the limits of the main area of injected pipes, there may be a broken border zone or detached neighbouring zones in

West Dome property; Left boundary of an école showing feldspar grains; cross section

which the rock, originally greenstone, has taken on a mottled appearance, and may contain a few sporadic pipes of quartz porphyry. The mottling on close inspection, appears to be due to ramifying bands and branching areas of extreme alteration

isolating patches of greater resistance. The alteration agencies having followed lines of least resistance, left the unattacked areas oriented parallel with the foliation of the rock. The alteration appears to consist in the greater development, or in the introduction of light coloured constituents, which on microscopic examination are found to consist chiefly of acid plagioclase and quartz. To a slight extent these minerals are developed to vary-

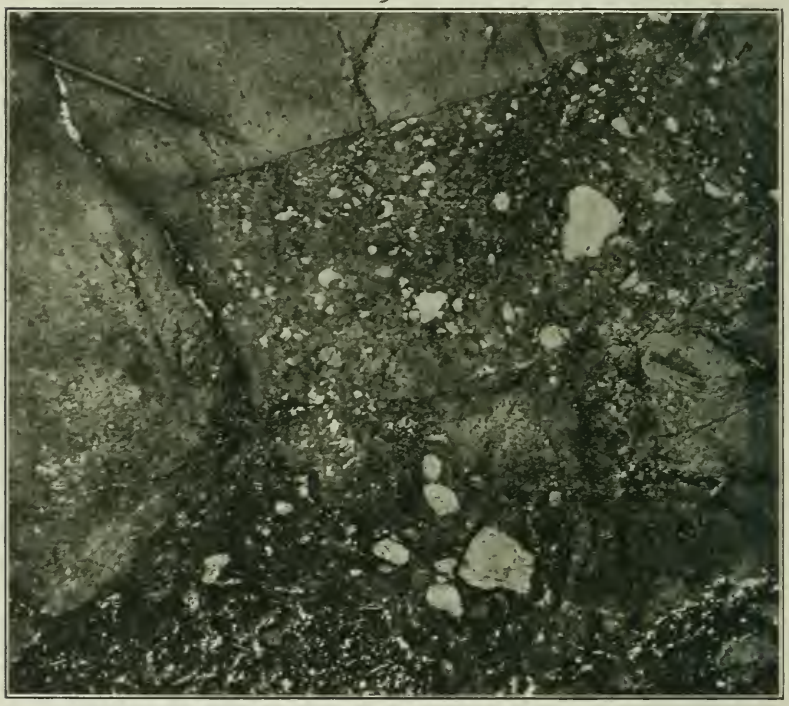
Bewick-Moreing Property
Excellent example of transition from an école into the typical border zone of transfusion

ing degrees in some of the island-like areas, but the phenomenon is most pronounced in the intersticial spaces.

Still beyond these areas similar zones are frequently found in which silicification is the conspicuous effect. And in a general sense it may be said that beyond the reach of silicification, carbonation is dominant and indicates the proximity of a quartz

porphyry mass, although properly, carbonation is not a part of the phenomena of *école injection*.

That the acid pipes are not squeezed remnants of such bands as characterize lit-par-lit injection is indicated by the complete isolation of large pipes in green matrix, and the occurrence of them in narrow tongues of the mottled alteration areas.



Bewick-Moreing Property
Excellent example to show limiting of école effects by old joint plains in greenstone

Wherever these phenomena exist, one may see scattered through the injected areas and particularly on and just beyond their margins, small white specks. These have proven to be crystals of acid plagioclase, and are regarded as having resulted from the transfusion of feldspathic materials from the nearby magna.

To offer a final explanation of the phenomena described is beyond the scope of the present paper. Microscopic studies are still under way, and it is hoped may make such a discussion possible in the future. However, on the basis of the field and

West Dome Property; Member of an école with inclusions of schistose greenstone; cross section

microscopic work already done, the writer ventures for the sake of completeness to suggest the following tentative explanations.

The general association of carbonation with masses of quartz porphyry, the distinct association of silicification with the neigh-

borhood and borders of *écoles* and the mottling effects between the silicified zones and the *écoles* themselves, are regarded as indicating the circulation of charged waters in these places simultaneously with the injection of the magna and emanating from its front. This idea is further borne out by the transfusion of magmatic materials into the border regions of the *écoles*.

The mode of injection of the pipes of quartz porphyry is the prime enigma. They are, as has been said, isolated, like the individuals of a school of fish; and cases are numerous to prove that they never had any other form, nor any better connections with each other, than appears at the present time. Intense squeezing cannot be employed to explain their isolation. Doubtless there has been compression, and the development of schist structure since the injection, and it must have produced lateral compression with accompanying longitudinal elongation; but such effects can only have intensified the flatness and separation which already existed among the pipes.

Illuminating evidence is found in the cases of certain large pipes which contain inclusions of the greenstone oriented in the usual direction. More is found in the nature of some of the light coloured areas in mottled zones, the microscope showing them to have the usual constituents of the quartz porphyry, but with more cases of granophyric intergrowth in the feldspars, of inclusions in them of zoisite needles, and in the greater abundance in the matrix, of chlorite, the pipe being intermediate in composition between the normal greenstone and the quartz porphyry.

The suggestions are strong of a metasomatic process of replacement of the greenstone by the acid pipes; but this critical matter cannot be further discussed as yet.

Prof. Frank D. Adams of McGill University in his studies of the phenomena accompanying the masses of granite gneiss injected into amphibolites and limestones of southern Ontario [3] calls attention to the parallelism of inclusions in the gneiss near the borders. In those cases he finds evidence of intense movement of the invading magma, along its margins, drawing out the inclusions into lenticular form, and their semi-assimilated remnants into long bands parallel with the direction of flow; and the mode of separating included fragments is thought to be mechani-

[3] "Geology of the Haliburton-Bancroft Area of Ontario", Memoir 6. Can. Geol. Sur.

Porcupine Crown; Typical Ecole

cal. In the case under discussion, however, no such flow structure has been recognized, and the separation of included fragments is thought to be partly chemical, that is, through the assimilation of the material by the magma. If the magma or its allied fluids had the power to assimilate the greenstone where it was simultaneously being replaced by the pipes of an *école*, it seems as if it should have the capacity to assimilate included fragments also, apparently separating them by making their boundaries retreat from one another through chemical absorption. Such processes would imply extreme slowness of flow and the presence in the advancing magma front of considerable water.

It has been shown that the magna was intruded into basic rocks already in a schistose condition. Doubtless they must have contained much chlorite and sericite, as at present, and other hydrous minerals. If these rocks had been pure chloritic schists they should have contained from 9% to 13% water of crystalization; but the chlorite being in only small proportion the percentage of such water must have been very small. However, the rocks have been carbonated and otherwise much altered since the injection occurred, which must have reduced the proportion of the micas. If ground water may be presumed to have filled the pores of the schist to the depth at which the intrusion occurred it must also be counted as an important source of water. At any rate, since the minerals of the fresh quartz porphyry were not hydrous, the water which must have been assimilated with the chloritic schists, if the magna stoped its way toward the surface as supposed, must have been in a free or uncombined state in the magma, especially at its active front. Whether or not juvenile water was present in the magma has not yet been determined; but this source of water, perhaps originally meteoric, is suggested as worthy of consideration.

SAFETY WORK AT THE CANADIAN COPPER COMPANY'S MINES AND SMELTER

By E. T. Corkill, Safety Engineer

Annual Meeting, Toronto, 1915.

In July 1913, the Canadian Copper Company organized a Department of Safety, for the purpose of accident prevention. The work of this department is largely in charge of the Safety Engineer, working in conjunction with the Central Safety Committee. This Committee consists of the President, General Superintendent, Chief Engineer, Superintendent of Mines, Smelter Superintendent, Chief Physician and Safety Engineer.

The General Superintendent is chairman of the Committee, and the Safety Engineer, secretary. Meetings are held during the first week of each month for consideration of the monthly accident report of the Safety Engineer, and recommendations brought forward for accident prevention. At these meetings the accidents occurring during the month preceding are discussed, and ways and means devised for the prevention of similar accidents wherever this is possible.

In addition to this Central Committee, Workmen's Safety Committees have been formed throughout the different departments. These committees consist of from five to ten men, depending on the size of the department, and meet every two weeks. The Safety Engineer meets with these committees and keeps a record of all their recommendations, advising them at a subsequent meeting as to the disposition of these recommendations. Since the organization of these committees many recommendations have been received and fully 90 per cent have been carried out. The work of these committees from the Smelter, Mechanical and Transportation Departments has been exceptionally good. The mines' committees have not been so successful owing to the preponderance of foreign labour. The duty of each committee is to take notes of defects in machinery, buildings, methods of working or handling material, or of any conditions throughout the works which may be the cause of accidents to employees.

Each member of the committee is allowed the necessary time from his work to attend these meetings, and if he is on another shift he is allowed three hours time for attending.

Reports are made out by the doctor for every accident case he attends, giving date, nature of accident, and probable period of disablement. This report is forwarded to the Safety Engineer. A notice is also sent by the doctor when the man is able to resume work, and a card is filled out and given to the injured person to be presented to foreman or timekeeper, giving the date the man is allowed to resume work.

A foreman's report is also made out and forwarded to the Safety Engineer, giving full particulars of each accident. In this manner a full and complete record is obtained of all accidents that necessitate a workman laying off work.

As an encouragement to the men, and to stimulate interest in the safety work, a pennant is awarded to the building at the smelter that has the lowest accident rate for the month. A pennant is also awarded the mine that has the lowest accident rate.

The fatal accident rate for 1914 was the lowest in the history of the Company. By serious accident is meant all accidents which incapacitated a workman for more than 35 days. This includes all fractures, and in the Mines Department for 1914, included a number of accidents where the workman had his finger or toe badly bruised while tramming. These injuries are often very slow in healing and on account of the nature of the work, the injured man is not allowed to return to work until his injuries are fully healed. There were no accidents during the year which caused permanent incapacitation. A comparison of the accident rate for 1914 with the year 1913, shows the following:— Number of fatalities per 1,000 men employed, .53 or decrease of 90 per cent over 1913. Number of serious accidents per 1,000 men employed, .19 or decrease of 19 per cent over 1913. Number of minor accidents per 1,000 men employed, .70, or decrease of two per cent over 1913.

All the different departments of the Company showed decreases in all classes of accidents with the exception of the Mines Department in the serious accidents, and as pointed out above, was due to the increase in seriously bruised fingers and toes among the trammers. The last half of the year has shown a

marked decrease in this class of accidents, due to change in underground conditions and to fewer men being employed in the older levels of the mines, where the clearances for the tram cars under the old drywalls and at chutes were not sufficient. The installation of the crusher on the sixth level and the putting in of an ore pass from the the fifth to the sixth levels at Creighton Mine, has eliminated the dumping of cars directly into skips and consequently the congestion of tram cars around the stations due to the waiting for skips. All the ore from the fifth and sixth levels at No. 2 shaft is now put through the crusher and into a storage bin from which the skips are loaded. This improvement has lowered the accident rate among the trammers very materially.

A comparison of the accidents that occurred in the various departments during 1914 with the year 1913, shows the following:

Mines Department:

Fatal accidents.....84 per cent decrease.
Serious " .....18 per cent increase.
Minor " .....Same.

Smelter Department:

Fatal accidents.....None, compared with three for 1913.
Serious " .....33 per cent decrease.
Minor " .....29 per cent "

Mechanical Department:

Fatal accidents.....None, same as 1913.
Serious " .....70 per cent decrease.
Minor " .....17 per cent "

Transportation Department:

Fatal accidents.....None, compared with 2 for 1913.
Serious " .....65 per cent decrease.
Minor " .....18 per cent "

Electrical Department:

Fatal accidents.....None, compared with one for 1913.
Serious " .....None, compared with 33 per 1,000 men employed for 1913.
Minor " .....55 per cent decrease.

As much attention should be paid to serious accidents, those causing a loss of time of thirty-five days or more or a permanent injury, as to fatal accidents. In fact, under the Workmen's Compensation Act it apparently costs less money in compensation if a man is killed than if he be permanently disabled. The cause producing a serious accident would have produced a fatal one with a few seconds or a few inches leeway.

The old excuse of the foreman that the man had been told not to go in a certain place or to do a thing a certain way has been worn threadbare and carries no weight. As I stated some years ago in a paper read before the Institute, "The lack of discipline in the mines of Ontario was responsible for the high accident rate." I still believe that to be true. Strict adherence to the orders of the foreman is absolutely necessary if we wish to reduce our accident rate. Of course, we must have good foremen. One of the main causes for our success in the reduction of accidents has been in requiring better discipline. Our one fatality during the year was due to a workman disobeying orders and resulted in the death of the shift boss who gave the orders. The shift boss himself was disobeying the orders of the mine superintendent and lost his life by doing so. Even with strict discipline a large percentage of our accidents during the year were caused by someone disobeying orders.

In our smelter during 1914 we had three serious accidents, which resulted in a broken leg, a broken arm, and the loss of one eye respectively. In all three cases either the injured man or a fellow workman disobeyed orders. In the mines over fifty per cent of the serious accidents were the result of disobedience or carelessness on the part of the injured party.

The cause of accidents in the Company mines during 1914, were as follows:—

Falls of rock	7 per cent.
Shaft accidents	2 " "
Explosives	2 " "
Tramming accidents	52 " "
Accidents at drill	14 " "
Miscellaneous	15 " "
Surface	8 " "

It will be seen that the big source of accidents is in tramming, the injuries consisting largely of bruised toes and fingers and are not usually serious.

The nature of injuries to Company employees during 1914 was as follows:—

Injuries to fingers or hand	42 per cent.
" toes or foot	20 " "
" leg	9 " "
" head	8 " "
" eyes	5 " "
Face burned	3 " "
Injuries to arm	3 " "
Body bruised	3 " "
Back sprained	2 " "
Miscellaneous injuries	5 " "

Comparison of the Canadian Copper Company's fatal accident record with Ontario and various metal mining states:—

Place	Total No. of employees	No. Fatalities	Tons ore produced per fatal accident	Fatal accidents per 1000 men employed
Canadian Copper Co....1914	1,883	1	682,084	.53
Ontario.....1913	16,000	64	.....	4.00
" 1912	15,108	43	.....	2.84
" 1911	14,543	49	.....	3.37
British Columbia.....1913	3,952	13	205,000	3.04
United States.....1913	193,088	683	.....	3.54
" " 1912	169,199	661	.....	3.91
Michigan Iron Mines..... (over a 10-year period)	152,430	716	148,745	4.69

The Economic Side of Accident Prevent'on.

There is absolutely no argument as to whether safety work pays from a humanitarian viewpoint. It pays, and pays enormously.

With the enactment of the Workmen's Compensation Laws in Ontario, British Columbia, and many of the States in the United States, the subject of cost of accidents has received very serious consideration. The records of a number of coal mining companies in the United States who have been carrying on safety work for a number of years, show that their costs for workmen's compensation have materially decreased.

The Workmen's Compensation Service Bureau of New York, which is the department for Accident Prevention, Inspection and Classification for 187 of the largest Employers' Liability Insurance Company reports as follows:—

"During the year 20,000 plants were inspected, comprising 1,200,000 employees and involving a payroll of over $700,000,000. Over $2,000,000 in compensation insurance premium was saved those employers through an actual reduction of the hazard in their plants."

The Canadian Copper Company has for over ten years carried no Employers' Liability Insurance, but has settled directly with the employee or his dependents. The cost of compensating all claims for 1914 was .64 per cent of the payroll. Of this amount .42 per cent was paid for non-fatal accidents and the balance for the fatal. The scale of compensation adopted by the Company in July, 1913, was 60 per cent of the employee's wages for the time disabled in non-fatal cases. This applied to all hospital cases and was paid without the employee making any claim.

The total cost to the Company by departments, was as follows:—

Mines	1.2 per cent of payroll
Smelter	.2 " "
Mechanical	.02 " "
Transportation	.3 " "
Electrical	Nil

The new Workmen's Compensation Board has fixed the following rates for 1915:—

Mines	3.0	per cent of payroll
Concentrators	.8	" "
Smelter	1.5	" "
Quarries	2.5	" "
Operation of electrical power plants	2.5	" "
Operation of railway	6.0	" "

All mines have been placed in one class and are assessed the same rate without regard to the safety precaution taken by the operators.

The same can be said of all the other classes of industries in the Province. This form of assessment is manifestly unfair to those operators who are spending thousands of dollars each year in making their plants safe and endeavouring to reduce their accidents, compared with those who are doing nothing.

ACCIDENT PREVENTION AT ONTARIO MINES

T. F. Sutherland, Chief Inspector of Mines, Toronto, Ont.

Annual Meeting, Toronto, 1915

During the past three years several mining companies in Ontario have, in an effort to reduce accidents, formed safety departments. In some cases this department is in charge of a safety engineer; in other cases a mine inspector is appointed who works in conjunction with the mine captain; while a third method is the appointment of safety committees. In all cases marked success in the reduction of accidents has resulted. The following is a brief description of the methods adopted by the different companies.

The safety departments of the Dome Mines Limited and the Mond Nickel Company having only been formed in the last two months of 1914, the details of their organization have not yet been worked out.

Beaver Mine, Cobalt

Mr. Frank Culver, president and general manager of this company, states that the idea of a mine having its own inspector was suggested to him by Mr. Thomas Jones, of the Buffalo mine, in a discussion regarding the prevention of accidents at mines. Mr. Jones intended to employ a mine inspector, and the idea so appealed to Mr. Culver that he immediately put a man on this duty and worked out the details later.

The inspector at the Beaver, in the first place, is responsible only to the superintendent. The underground foremen have no authority over him whatsoever. He inspects the mine twice a day and fills out an inspection report for each shift. This report is signed by the outgoing and incoming shift boss, as well as by the mine captain, and goes to the superintendent. Should this report show that certain conditions underground are unsatisfactory, the underground foremen will certainly rectify matters, so that they will not have to sign a second unsatisfactory report and have it go to the superintendent.

For repairs that are required the inspector fills in a second report form in triplicate. One copy is handed to whoever is responsible for making the repairs; when repairs are completed it is signed by this person and forwarded to the superintendent's office. All miners going on and off shift report at the inspector's office; they report any missed holes. Miners fire their own missed holes. When danger exists, such as a missed hole, bad ground, etc., a red flag is stuck up. Every man, no matter of what nationality, recognizes immediately that something is wrong. This does away with the danger of a verbal warning not being understood, and is one of the best features of the whole system of accident prevention. Regarding this feature the *Engineering and Mining Journal* of February 31st, 1914, says: "Intelligent initiative like this is only what might be expected from the Beaver, operating as it does under probably the most advanced and effective inspection system in the world."

The inspector's report, bringing to the attention of those responsible any repairs needed, also tends to greater efficiency. If a machine is out of order, chute blasted, pipe line broken, etc., repairs are immediately made, and men do not lose a couple of hours in getting to work.

Buffalo Mine, Cobalt

In December, 1911, The Buffalo Mines, Limited, established an inspection department in connection with all underground work in their mining operations at Cobalt. An inspector was appointed, with an office close to the chief exit from the mine, where the miners coming off and on shift could report. As each shift comes off the miners report holes blasted and any missed or cut-off holes.

In the interval between shifts the inspector makes a complete tour of the mine and inspects each working. Any missed or cut-off holes discovered are indicated by a red flag left for that purpose, and are also reported to the on-coming shift with instructions to blast the flagged holes immediately and bring the flag back to the inspector's office. If missed or cut-off holes are covered by muck, a flag is left stuck in the muck pile to warn the muckers not to use a pick. When any loose

ground or broken timber is discovered, or muck hung up in stopes, a flag is left to mark the spot, and the scaling and timber gangs are instructed, when coming on shift, to make everything safe and return the flag to the inspector's office before proceeding with any further work. This system of warning by red flags eliminates any danger of verbal instructions or posted regulations being misunderstood, or being unintelligible to foreigners; a miner of whatever nationality would understand a red flag to mean danger.

All miners are instructed as to the law and the penalties incurred in connection with starting holes in old bottoms, leaving gates or guard-rails open, or pulling signal cords above or below where the cage is standing, and notices are also posted to this effect.

In addition to an inspection of the work between each shift, the mine inspector makes a general tour of the whole mine once a week, making a complete inspection of all hoists, cables, signals, ladders, timber, manways, cars, cages, tracks, pumps, machines, air lines and powder magazines. The powder magazines are inspected twice daily and temperature reported. No powder is allowed to be left in any place of work, and the powder man is instructed to bring back all powder not used, special attention being given to the inspection of the air courses and ventilation.

Every man going on duty passes under the inspector's observation, and no man would be permitted by him to go underground in an unfit condition for work.

Economy is effected in the time saved, due to miners and shift bosses receiving a report of the conditions existing at their place of work before going underground; thus needed repairs are made immediately and dangerous conditions rectified before proceeding further.

Efficiency is increased as all are aware that no lax conditions will be allowed to exist. The rigid inspection twice daily of all working places keeps the men up to a better standard of efficiency, and attention to regulations than anything else could.

The dominant feature of the system is that carelessness

will not be permitted, and when carelessness is overcome a large factor of danger is eliminated.

Cobalt Lake Mine, Cobalt

At the Cobalt Lake mine, in July 1914, a Safety Committe was formed amongst the foremen and men. The manager, Mr. M. B. R. Gordon, in investigating several minor accidents found that the majority of these accidents might easily have been avoided and were due to the negligence of the employees. The ordinary precautions against accident were being observed, but instances were continually occurring where the indifference or thoughtlessness of the employee was endangering his own safety or that of a fellow workman. Thus neglect to immediately repair a broken rung in a ladder, carelessness in scaling a small rock or cleaning off landings, etc., a workman sledging besides a trammer in a drift, broken board walks and nails left sticking through boards, tramming loaded cars too close to one another, and many other thoughtless practices, were the causes of numerous minor accidents. It was decided that the only way to overcome these conditions was to enlist the help of the men themselves. Consequently a safety Committee was formed with the manager as chairman. The different shift bosses were to be the heads of different committees on different work; each shift boss and foreman was to have two employees from their particular crews elected by the men themselves, and associated with him; the mill superintendent was to be vice-chairman, and from the shifts in the mill and machine shop a similar method was to be followed as in the mine. The two delegates to the Committee from each shift, elected by the men, were, between the intervals of the different meetings, to gather such information by themselves and from other employees, as they deemed of interest to the main committee. When any dangerous point was located it was to be at once reported to the head of the committee, and, depending upon the seriousness of the situation, passed on up to the foreman or sent direct to the chairman. But all these improvements, or places of danger, or accidents that had occurred, were to be reported at the next meeting for discussion by the other members.

By this method a considerable number of the most intelligent men on the property in all the different branches of the work were discussing every fortnight among themselves and with the manager the practices and conditions which appeared to them unsafe. This discussion invariably led to suggestions for improvements in methods, and the Safety Committee became also an Efficiency Committee.

Regarding the results Mr. Gordon states: "The results obtained more than justified the trouble taken, for not only did these minor accidents practically cease, but we also obtained in a large degree much greater efficiency from our employees and I can state definitely that it is a saving of time and money both to the employer and the employee."

Canadian Copper Company

Acccident prevention by the Canadian Copper Company is dealt with in a separate paper by their Safety Engineer, Mr. E. T. Corkill.

Dome Mines, Porcupine

During the lattter part of 1914 a trained mechanical engineer was appointed Safety Engineer at the Dome Mine and a Committe on Safety, Sanitation and Efficiency was formed. The Committee consists of the heads of the operating departments and meets once a month. The Safety Engineer makes a daily inspection of the various parts of the mine and plants, and sees that all safety recommendations made to department superintendents are complied with.

General Chemical Company, Sulphide

The General Chemical Company, of which the Nichols Chemical Company, Sulphide, Ontario, is a subsidiary, employs 3,000 men in its various mines and chemical plants throughout the United States and Canada. In Ontario the chemical works and mine at Sulphide, in Hastings county, is operated by this company. The manufacture of chemicals in a modern plant under expert supervision is not essentially a dangerous

occupation, nevertheless a large number of serious, non-fatal accidents indicated to the company the need of a "safety-first" campaign among all its employees. Accordingly, what is known as a "Central Service Committee" was established at the New York office in charge of Mr. Robert K. Painter, mining engineer. Acting with Mr. Painter are Mr. Black, sanitary engineer; Dr. Talbot, consulting engineer; Mr. J. R. Bueno, secretary and general manager; Mr. W. H. Nichols, Jr., member *ex officio*. The function of this Service Committee is the supervision and study of all matters pertaining to safety and sanitation throughout the company, at all their works, including also welfare work among the employees. Bulletins are issued monthly by this Committee, containing extracts from papers read before conventions and societies regarding safety methods and devices. Latterly, also, this bulletin has included a comparative statement showing all accidents at the different plants; copies are distributed anong the employees and those interested.

At the Sulphide plant, under the Superintendent, Mr. W. H. DeBlois, there has been established what is known as the Works Safety Committee, consisting of one employee from each department of the plant, the mine itself having a separate and similar organization. The departments represented are as follows:—

1. Rock house at mine.
2. Mechanical department.
3. Burner and crusher department.
4. Sulphuric acid department.
5. Nitric and hydrochloric acid department.

Each member inspects the whole works in turn and reports to the superintendent on a form provided for that purpose. The different suggestions are considered by the superintendent and the foreman of the department interested, and those approved are ordered to be carried out at once. The form has space at the bottom for the signatures of foremen certifying to work being completed as directed. The committee also investigates and reports on any important accident.

A first-aid equipment, including pulmoter and surgical instruments, is on hand for immediate use, and is in charge of

the Chief Chemist, Mr. Jordan. The company's physician visits the plant regularly, and under ordinary conditions can reach the plant in twenty minutes if required.

At various points about the works are posted photographs published and distributed by the National Council for Industrial Safety, Chicago. One card, for example, shows a workman's hand swollen and permanently deformed as a result of blood poisoning from a neglected scratch on the finger. Beneath this photograph is printed the following warning: "This emphasizes the necessity for impressing on workmen that they must tell their foremen whenever thay are hurt, even if it be only a scratch, so that proper means may be taken to prevent complications."

MOND NICKEL COMPANY

In the latter part of 1914 a Safety Engineer was appointed by the Mond Nickel Company with headquarters at the Garson mine, that being the property which had the highest accident rate. Two rule books have been published by the Company, one for the superintendents and foremen, the other for employees at mines.

NIPISSING MINING COMPANY

The Nipissing Mining Company have prepared an excellent rule book which is published in English, French and Finnish. A copy of this rule book is given to every person who becomes an employee of the Company and a receipt taken, thus impressing on the employee the necessity for reading and observing the rules of the Company.

THE STEEL COMPANY OF CANADA, LIMITED

Realizing the necessity of a "safety-first" organization among its employees, the above Company issued on January 1st, 1913, a book of rules covering all departments, a copy of which is given to every employee. These rules were formulated by a committee of departmental heads, and the penalties for infraction are very drastic. Every foreman is responsible for

the instruction of the men in his charge in these regulations, and a proper understanding of the same by them. What might be termed an interlocking system of reports from one department to the other is enforced, whereby a workman from one department ordered to work in another must first report to the head of that department before starting work. For example, a machinist from the mechanical department is ordered to make certain repairs on the crane track. He must first report to the chief electrician, who will send an electrician with him to see that he does not come in contact with high tension wires, etc.

The introduction to the book of rules includes a letter from the General Manager, Mr. Hobson, from which the following extracts are taken:—

"The prevention of accidents to its employees is the earnest desire of The Steel Company of Canada, Limited. While the company may provide all possible safety appliances, accidents will be of more or less frequent occurrence if employees do not assist in their prevention by cultivating habits of caution. For the guidance of employees this set of rules, regulations and suggestions is issued, and it is the earnest wish of the company that they be carefully studied and lived up to. Not only is it the duty of every employee to individually exercise care and caution, but it is his duty to see that his fellow workman does not violate any rule. All the safety appliances in the world will not prevent accidents if every man is not on the watch to guard against them."

The following memorandum has been issued recently to the heads of departments:—

1. Safety committees must be at once formed in each department.
2. Each committee will consist of three employees, who will act for three months.
3. The committees will make regular inspections during the company's time, of plant, machinery, tracks, etc., belonging to their respective departments.
4. Report to the Central Committee any changes which they consider will add to the safety of employees.

5. These changes will be considered by the Central Committee, who will decide as to whether they shall be adopted or not.
6. The Central Committee to consist of the manager of works, the superintendent or head foreman, together with one other person selected by the manager.
7. The Safety Committees should make the inspection of their departments at least once a month and report on general conditions of departments, recommend changes where safety could be promoted, and make suggestions on accidents which have happened during the past month.
8. Cleanliness and order should always be considered as an essential of safety, and department committee should carefully take note of places where improvements could be made along these lines.
9. I am sending you herewith blue print of safety signs, which may be had from our Purchasing Department, and which you will please hand to your safety Committee, who will see that they are placed at all danger points. Plain enamelled signs can also be supplied.
10. Goggles for the protection of eyes will be supplied, and when men refuse or neglect to use them when engaged on work where there is danger of eyes being injured, they should be at once discharged.
12. Any employee who is careless with regard to his own safety or the safety of his fellow workmen, or who is known to be of intemperate habits, must be dismissed.

Dr. Talbot, of the General Chemical Company, says that a distinction must be made between "first-aid treatment" and "emergency treatment." There are instances on record where first aid has been given that has not been aid but an injury, and the company has had to pay for it. In other words, first aid should never take the place of skilled aid. Yet real first aid oftentimes will make the necessity of skilled aid less expensive and less necessary. First aid is merely what can be done to avoid the necessity for skilled aid. In emergency treatment, however, we have something entirely different. In every

works we should not only have first-aid men and understudies for first-aid men, but there should be somebody, by night as well as by day, always available to give emergency treatment also.

The Steel Company of Canada made this distinction between first aid and emergency aid nearly two years ago, and built an emergency hospital at the works, in charge of Sergeant Matthews, an English Army non-commission officer, retired after long service in India and Africa. He is on duty night and day at the hospital, and his records show that for the six months ending April 30th, 1914, 432 cases were given emergency treatment. The majority of these cases were bruised hands and feet, cuts and burns, minor injuries to eyes, several gassing cases from the furnaces. Sergeant Matthews organized first-aid classes throughout the works, and now has fifty-two thoroughly instructed young men scattered throughout the various plants, twenty-eight of whom are at the steel plant. Most of these have passed second and third-year lectures and have been given certificates. Three of them have taken the full course at the Halifax Military Hospital, their course being paid for by the Militia Department.

THE INSTRUCTION, ORGANIZATION, AND CARE OF RESCUE PARTIES USING SELF-CONTAINED BREATHING-APPARATUS IN MINES AFTER AN EXPLOSION, OR IN FIGHTING MINE FIRES

By George O'Brien, Instructor of the Provincial Government Mine-Rescue Station, at Fernie, B.C.

Western Branch Meeting, March, 1915.

The value of self-contained breathing apparatus used in connection with exploratory and recovery work in mines after an explosion or mine fire is now beyond all doubt, so it needs little recommendation to the modern mining man.

It is not the intention of the writer to select any particular type of apparatus and devote his remarks exclusively to it, for there will always be some difference of opinion with regard to the different types of apparatus now being manufactured. He proposes, rather, to deal with the instruction, organization, and care of rescue parties, and he feels confident that if the following instructions are carried out as nearly as circumstances will permit, the dangers attending exploratory and recovery work will be greatly minimized. It has been proven, conclusively, that in most cases where members of rescue parties have lost their lives, some important detail has been overlooked, or neglected; therefore, the writer sincerely hopes that this small effort will be of some assistance to those engaged in the hazardous work of exploring and recovering a mine after one of those terrible catastrophies incident to coal mining has occurred.

Instruction

The instruction of men in the use of mine-rescue apparatus is of the greatest importance. After selecting some particular type of apparatus (of which there are a number now being manufactured) a competent instructor should be employed for the purpose of instructing and training men in its use. A practical mining man should, if possible, be secured as instructor, and one

who has had a technical training in mining, for greater efficiency can be obtained by having an instructor who can give lectures on the different mine gases, their behaviour in mines, and their effect on human life, etc., etc., as this is a very important feature of the instruction, and in the writer's opinion no instruction is complete without it.

A fairly large building should be secured for training purposes, this to embrace a smoke-chamber say 18′ x 30′ x 12′, a work-room 12′ x 16′, and a lavatory or wash-room.

The smoke-chamber should contain a number of obstructions, to imitate as nearly as possible the conditions one would expect to find in a mine after an explosion.

The workroom should contain a work bench and tools, also, for the apparatus, lockers or cupboards, which should be mice and rat-proof as these animals will, in a very short time, destroy the breathing-bags and connecting tubes if they get at them.

The wash-room should contain a few wash-basins, also two sinks. The sinks are necessary for the purpose of washing the breathing bags each time they are used. This is very necessary for sanitary purposes. A few lockers should be provided for the men's clothes.

A good schedule of work should next be arranged, and when once adopted should be carried out to the letter. The training should extend over a period of twelve consecutive days, and at least twenty hours should be spent in the smoke room doing actual work. All men should be medically examined and pronounced fit for the heavy work necessary in rescue operations. If proper instructions shall have been given during this period, any ordinary miner should be able to pass the required examination at the end of the session.

It is very necessary that instruction in resuscitation be given during the period of training. This can be accomplished by instruction in the use of the pulmotor and inhalator. A few lessons should be sufficient, as the operation of the pulmotor and inhalator is very simple, and the ordinary workman should be able to operate them after a few demonstrations by the instructor.

Some of the most important features of the instruction should be: The composition of the atmosphere, the phenomena of breath-

ing, the quantity of oxygen required to sustain life, the effect and behaviour of the different mine gases, where found, how produced and the chemical re-action that takes place after an explosion of fire damp or coal dust, or both combined. Each man should thoroughly understand the apparatus and be able to assemble it, and test it, before being allowed to act as one of a rescue party.

After the training has been completed at the Central Station, periodical training for each man should be carried on at the different collieries, say once a month. This will keep the men in touch with the apparatus, and any improvements or additions that shall have been made to it.

In the writer's opinion, a very good plan would be to have every trained man made acquainted at least, with the main airways in the different mines at each colliery, as valuable time is sometimes lost when rescue parties have to depend entirely on blue-prints to follow the different roadways. Blueprints are valuable in most cases, but are often misleading in a mine after an explosion has happened. There may be shown on the blueprint a number of places which cannot be recognized in the mine, through being walled off or concealed by timbering; or possibly caves may have occurred and obscured them. The physical features of the average mine are so badly disfigured after an explosion, that persons well acquainted with the mine workings prior to the explosion often have great difficulty in recognizing certain parts of the mine after the explosion. This is a well known fact and should be guarded against, since every second counts in the saving of life after a disaster, speed in getting to the entombed miners being all important.

ORGANIZATION

The greatest factor in actual mine rescue work is the organization of the different parties. A well-organized rescue party should consist of 54 men divided into 9 crews of 6 men to a crew, one man in each crew to be the leader or captain. The whole party of 54 men should then be divided into reliefs of 18 men to the relief. This arrangement will allow a working shift of 18 men two hours work and four hours rest. The first point to be determined is where the base of supplies is to be located.

In most mines after an explosion has occurred there is a point where the danger and safety zones meet. If this point is some considerable distance in the mine, the base of supplies should be located there; if it is near the surface, the base should be located on the surface, and if a drift or slope mine, a safe distance from the mine mouth and to the side of it, so as not to be in the line with the mine mouth in case of a second explosion occurring, as is sometimes the case. Having arranged the different shifts and reliefs, a reliable and competent person should be stationed near the mine mouth for the purpose of checking each man in and out of the mine. This is absolutely necessary, for it gives the officials in charge of the organization a full knowledge of the length of time each party has been inside, and when to expect its return. No unauthorized person should be allowed to enter the mine, and perfect organization, and discipline, should be maintained. Before entering the mine, each man should see that his apparatus is in good working order. It should be thoroughly tested, either under his supervision, or by himself, as everything depends on the apparatus being in a safe working condition, and when the men see the apparatus tested they have much more confidence in it.

After establishing the base of supplies, two of the three rescue parties should enter the mine and, proceeding carefully, endeavour to ascertain if there are any of the victims alive in the mine; the third crew equipped with the apparatus but not using oxygen, remains at the base, ready at a moment's notice to proceed inside in case they are required for any emergency that may arise.

Some comment is necessary here as to the advisability of restoring the ventilation before exploration has taken place. There is much divided opinion regarding this feature of mine-rescue work. There is always the chance of smouldering fires existing after the explosion, also large volumes of unexploded CH_4 and CO, which may be driven on the fires and cause a second explosion. Another danger to be guarded against is the driving of poisonous or suffocating gases ahead of the ventilation, which, in the absence of fire, would surely cause death to any of the victims in its path, who may have escaped the first explosion, so that there are two dangers to be guarded against in the hasty

restoration of the ventilation. Opinions will possibly continue to differ on this point. However, the writer suggests that to explore first and then restore the ventilation after ascertaining that no danger of sacrificing further lives exists, is the safest method.

To go back to the original scheme of exploration. As the first party advances in to the workings, the position and situation of any bodies found should be noted. These bodies could be removed later, but no time should be spent in removing them at this stage of the exploration. The chief thing is to save those who have been fortunate enough to be alive; the dead can be removed later. Great care should be taken by the leading party in advancing, and the condition of the roofs and sides, and timbering should be specially noted. There is always the chance of a large cave occurring and cutting the exploration party off from its base. No advance should be made beyond any point considered dangerous from this cause until the danger shall have been eliminated. Members of rescue and exploration parties must remember that if, unfortunately, any of the party is killed, it causes great consternation throughout the whole of the organization; therefore, no chances should be taken.

The work of the second party should be to follow up and act as a support to the first party, assisting it in every possible way. A line of communication could, in this manner, be kept up with the reserve party at the base of supplies.

Every man should frequently examine his pressure gauge for the pressure in his cylinder determines the length of time he can remain in a dangerous atmosphere. It also determines the time for withdrawal of part, or possibly, the whole of the party.

In case any member of the rescue party shall become weakened, the leader should at once have him removed to the base, and should send at least three men to assist him out. The reason for this is obvious. An already weakened man may possibly get weaker, and if alone, may collapse on the road, so his life would more than likely be lost, but if accompanied by three, or more, men the chances are that nothing of this nature will occur.

If the party is thoroughly well organized there should be no

danger of any member of the party losing his life by being overcome by weakness, or by an inhalation of the afterdamp.

In the event of any member of the party being overcome, he should not be allowed to re-enter the mine for a period of, say, twelve hours after his recovery, especially if the cause of collapse be due to an inhalation of carbon monoxide, as it takes considerable time to purge the blood of this gas, and a person whose blood is partly saturated with CO is much more liable to collapse than one in normal condition. The leader, or captain, of each party is responsible for the safety of his men, and, on no account should he allow them to over-exert themselves while wearing the apparatus, as this is a frequent cause of collapse.

A few hundred feet of one inch hemp rope should be handy in case it becomes necessary to drag a member, or members, of the party to safety, as sometimes this may have to be done.

The chief thing for all members of rescue parties to remember is the fact that the breathing apparatus they wear is only a machine and that great care must be taken of it while in a dangerous atmosphere. They should also remember that no good can be accomplished by rushing things too much. Under no conditions should members of a rescue party become separated; all should remain within touching distance of one another. No advance should be made by the leader unless he is absolutely sure that the whole party are together. It is quite possible that the last member of a party may get some part of his apparatus or clothing caught in passing over a cave, in getting by wrecked mine cars, or some other obstruction, resulting in his being left behind some distance before his absence is discovered. This can only be guarded against by the leader not advancing unless he is sure that all of his party are ready to advance with him. Similar care should be taken in retiring from a danger zone. No member of a rescue party should remain in a dangerous atmosphere unless he is confident that his apparatus is working properly. The writer feels that if these instructions are followed. a considerable amount of effective work can be accomplished.

Care of Rescue Parties

For a certain period of time following one of the dreadful coal mine disasters, very little thought is given to anything

excepting the recovery of the victims, and in most cases, the care of the rescue parties is entirely overlooked. This should not be, as the greatest efficiency cannot be obtained unless the rescue parties are properly looked after. The writer suggests that some building near the mine be equipped with a dozen camp cots, blankets, and mattresses, so that the men constituting rescue parties on returning from the mine could lie down and rest for a few hours. If a building be not available, a good-sized tent, say 16 x 24 feet, or two smaller ones, should be kept at every colliery for this purpose. A doctor, or doctors, should be in attendance, and if time permits the rescue parties should be examined and the conditions of their hearts and lungs ascertained before being allowed to wear the apparatus in a dangerous atmosphere. In any case, this should be done before they enter a second time. Any man having a weak heart, or suffering from a bronchial affection, should not be permitted to wear the breathing apparatus. No excitable person should be allowed to be a member, for he would be a danger to the whole party.

Provision should be made for a liberal supply of hot tea, coffee and sandwiches. A number of men should be detailed for this work. A caller should also be detailed, and be on duty for the purpose of getting the rested crews ready when the working crews are to be relieved. If this be done, there will be no suspension of the recovery work as is too often the case.

Rescue parties should not be required or allowed to do any unnecessary labour, such as carrying bodies to the morgue, or conveying supplies to the mine, etc., as all their strength will be required when their turn shall come for entering the mine with the apparatus. There are usually plenty of men available for the other work.

In conclusion, the writer once more wishes to impress on leaders and individual members of rescue parties, the absolute necessity for their conserving their strength, and of avoiding overexertion and excitement under the difficult conditions of rescue work. If this be done, the writer feels that his small effort to minimize the dangers attending the exploration and recovery of a mine after an explosion will not have been in vain.

NOTES ON A CROSSCUT ADIT DRIVEN BY THE PORTLAND CANAL TUNNELS, LIMITED

By W. J. Elmendorff, Seattle, Wash.

Western Branch, March Meeting

The main working of this company consists of an adit 3671 ft. in length; it crosscuts what is known as the fissure zone of Glacier Creek. Its objects are to provide an entry at depth to several adjoining mining properties, as well as drainage and transportation for them. The developing of a water power is also contemplated.

The portal of the adit is about $3\frac{1}{2}$ miles north from Stewart, B.C., and 350 ft. above mean high tide at that point. It is on the line of the Glacier Creek spur from the Canadian and Northeastern Railway. Work was begun in October, 1912, and crosscutting, or drifting, or both was continuously prosecuted from then until January, 1915. The power plant of the Portland Canal Mining Co. was leased in connection with the work.

The heading, $7\frac{1}{2}$ by 8 ft. in the clear, was driven a distance of 2916 ft. at an average rate of 8 ft. per day. The best weekly record was 67 ft., and the best monthly 271 ft. Two 8-hour shifts per day were employed, consisting of four machine-men and three shovelers. After the 1000-ft. mark was passed, four shovelers were employed. Compressed air at 100 lb. pressure was supplied by a water-driven, 520-cu. ft., Rand class D-2 compressor, and delivered into the tunnel through a 4-in. pipe. Two $3\frac{1}{4}$-in. Ingersoll-Rand E44 drills, mounted on a 7-ft. single screw bar were used. Miners and shovelers started work together. The bar was set up above the rock pile, and from ten to twelve holes were drilled from this position. By this time the rock was out, the bar was lowered, and three lifters were drilled. The first 1600 ft. of the tunnel is in greenstone, while the remainder is in argillites, quartz, and porphyry. With but two exceptions, during the entire work, a round was drilled in every shift. Five-foot rounds were drilled, but seldom 'bottomed.' Three 20-cu. ft. cars were used, and tramming was done by the shovelers. The track, of 20-lb. rails, 18-in. gauge, and $\frac{1}{2}\%$

grade, was laid on the left-hand side of the tunnel. One switch, about half way in, was used; the empty incoming car was taken off the track near the face to allow the loaded one to pass. At a point 730 ft. from the portal an 80-ft. adit was run, opening into a canyon. This not only made an excellent dumping place, but shortened the tramming distance by 750 ft. The shovelers laid and leveled the track. The floor from within 3 ft. of the face to 40 ft. back was covered with steel plates before blasting. A drain 12-in. deep was made on the right side of the tunnel, the lifter being drilled low for this purpose.

Ventilation was provided by two Schutte & Koerting blowers. The ventilating line consisted of 10-in. diameter, 24-gauge galvanized pipe. The pipe was made in 10-ft. lengths, with riveted joints, painted with tar and wrapped with muslin. The compressor was run two hours after blasting to clear the tunnel.

TIME AND COST OF OPERATIONS.

AVERAGE TIME OF OPERATIONS

	Hr.	Min.
Picking down—setting up	1	00
Drilling from upper set—up	4	30
Shoveling back—lowering bar	0	30
Drilling from lower set—up	1	15
Tearing down	0	10
Blowing out holes	0	5
Loading	0	15
TOTAL	7	45

DETAILED SUMMARY OF COSTS

	Total Cost	Cost per Foot
Labour:		
Machine men	$11,526.00	$3.953
Shovelers	10,711.00	3.673
Timbering	753.17	0.258
Drainage	123.00	0.042
Ventilation	224.50	0.077
Ties and wedges	394.60	0.135
Compressed air line	210.00	0.072
Compressor men	3,398.35	1.166
Blacksmith and helper	3,546.50	1.216
Superintendent	2,554.00	0.876
TOTAL	$33,441.12	$11.465

	Total Cost	Cost per Foot
Supplies:		
Powder	$7,469.25	$2.562
Fuse	391.50	0.134
Caps	92.00	0.032
Candles	421.90	0.145
Blacksmith coal	364.50	0.125
Machine drills and parts	1,464.77	0.503
Drill steel	343.57	0.118
Timbers	150.00	0.052
Repairs on compressor	77.40	0.027
Oils	97.00	0.031
Miscellaneous	216.00	0.073
TOTAL	$11,087.89	$3.802

	Total Cost	Cost per Foot
Equipment:		
Muck sheets	$ 110.00	.....
Steel rails	1,290.00	.....
Ventilators	225.00	.....
Ventilating pipe	960.00	.....
Compressed air line	1,070.00	.....
Cars	400.00	.....
TOTAL	$4,055.00	$1.391

Length of tunnel	2916 ft.
Total cost	$48,584.01
Cost per foot	$16.661

All supplies were bought at Vancouver, British Columbia, and sent north by steamship. The prices of some of the more important supplies at the tunnel were as follows:

Dynamite, 40%	$0.134 per pound
" 60%	0.159 " "
(powder was used in about the proportion of 30% of the former to 70% of the latter).	
Steel rails, 20-lb	72.24 per long ton.
Blacksmith coal	27.00 per ton.

The scale of wages was as follows:

Machine men	$4
Shovelers	4
Blacksmith	5
Compressor men	5

Beyond the point to which these figures apply the size of the tunnel was cut down, one drill instead of two drills being worked in the face and the costs per foot from then on decreased about $1 per foot. The principal ore bearing veins in the fissure zone were reached by the crosscut at 2364 ft., 2642 ft., 2916 ft., and 3560 ft., from the portal. All are similar in character, quartz breccia in argillites containing pyrite, sphalerite and galena, abundant in the order named, as their ores. About 2000 ft. of drifting has been done on these veins at a cost of something less than $16.00 per foot. In the last vein reached an orebody consisting of a system of ore seams was opened for a width of more than 30 ft.

Drifts north and south on the strongest of the ore seams in this vein and for a length of 300 ft., along it show an average width of 20 in. of ore and an average value of $11.90 a ton in gold, silver, lead, and zinc. Excluding the zinc the average value is about $10 per ton. Further drifting on this and some of the other veins is contemplated for the further development of this property.

COAL TAR PRODUCTS AND ARTIFICIAL DYESTUFFS INDUSTRY

By Lionel O. P. Walsh, Sydney, N.S.

Annual Meeting, Toronto, March, 1915

The fact that we have been almost entirely dependent on Germany for our supplies of dyestuffs, pharmaceutical and fine chemicals, has been brought home to us very forcibly during the last few months. The shortage of these products (and we are given to understand the stocks in Great Britain, Canada and the United States only represent two to three months' supplies), has compelled the industries concerned to seriously consider the possibility of supplying their own demands by manufacturing dye stuffs and pharmaceutical products in this country. The principal source of the raw materials is from coal tar, and a short description of what these raw materials are, how they are produced, and their relation to the dyestuffs and pharmaceutical products, may be of some interest.

Coal tar is a by-product obtained in the distillation of coal. The chief source is from the coke ovens engaged in the manufacture of coke for blast furnace and foundry use. The volatile matters distilling from the coal are led away through cooling towers and scrubbers where the tar together with ammoniacal liquors is condensed and washed from the gases which are carried on and used for heating, lighting or producing power. The condensed tar is allowed to stand in large storage tanks where it separates from the ammoniacal liquor and is then sold to the tar distiller. The ammoniacal liquor is distilled with lime, thus liberating ammonia which is combined with sulphuric acid, forming sulphate of ammonia, a very valuable fertilizer. Coal tar is a black viscid liquid having a specific gravity of 1.1 and 1.2. It is a complex mixture of compounds a great number of which have been insolated and examined, but the ones which have value and interest to the dyeing and chemical industries are benzol, toluol, solvent naptha, carbolic acid, cresylic acid, naphthalene and

anthracene. The proportions of these constituents vary greatly with the kind of coal employed and with the temperature at which the distillation is carried out.

In order to isolate these valuable products the tar is first submitted to a preliminary distillation from large wrought iron stills by which it is separated into the following portions:—

A First runnings,
B Light oils,
C Carbolic oils,
D Heavy Creosote oils,
E Anthracene oils,

whilst a residue of pitch remains behind in the still. From the first runnings and light oils, the benzol, tuluol, and xylol are obtained; from the carbolic oils the carbolic acid, cresylic acid and naphthalene and from the anthracene oils, the anthracene.

In the further working up of the first runnings and light oils these products are submitted to a fractional distillation by means of which a low boiling portion and a high boiling portion are removed. The middle portion is then subjected to a series of treatments with caustic soda, sulphuric acid and water. The resulting product is then again submitted to fractional distillation and from the crude benzol thus obtained, benzol, toluol and xylol are separated by rectification in a still of peculiar construction in which the vapor is caused to pass through a long column divided into a series of chambers by perforated plates in which the higher boiling portions are condensed and return to the still. The amount of benzol obtained from coal tar is comparatively small as most of it is contained in the coke oven gases after the tar is separated therefrom. This benzol can be extracted from the gases by mechanical 'scrubbing' with creosote oil, which becomes saturated with benzol. This creosote oil is distilled and the benzol thus obtained is treated in a similar manner to the first runnings and light oil.

Carbolic acid and cresylic acids are extracted from the carbolic oils by agitation with caustic soda, the aqueous caustic soda liquor is separated from the oil and after undergoing a purification is neutralized by acid—a mixture of crude carbolic

and cresylic acids separate. These are submitted to a series of fractional distillations and from the middle portions thus obtained the pure carbolic acid is crystallized out, the remaining cresylic acid is the liquid portion.

Naphthalene is obtained from the carbolic oils from which the carbolic acid has been extracted. On standing and cooling, large quantities crystallize out which are separated by centrifugation and pressing. The crude naphthalene is treated with caustic soda and strong sulphuric acid, and is frequently given other chemical treatment to remove impurities, after which it is finally distilled or sublimed. Thus obtained the naphthalene is almost pure.

The last portion of the coal tar distillation, the socalled 'anthracene oil,' is a thick, buttery, greenish mass. For the extraction of anthracene the oil is cooled and the anthracene which separates out mixed with other compounds is freed from the oily liquors by pressing and centrifugating. The greenish mass thus obtained, which only contains from 10 to 12 per cent of pure anthracene, is submitted to a long series of chemical purifications before the final product is raised to a commercial pure standard. From these so-called 'raw products' are manufactured the many hundreds of intermediate compounds which are necessary for the formation of dyestuffs. It must not be imagined that the manufacture of these compounds and the dyestuffs is an easy matter. The employment of numerous and large quantities of chemicals, such as nitric acid, sulphuric acid, etc., etc., is required; long and often tedious processes are involved and a highly scientific supervision is essential.

From benzol by treatment with nitric acid and a subsequent operation aniline is obtained. Aniline and compounds of analine are worked up into an enormous number of dyes of every shade known as the aniline dyes, of which methyl violet, malachite green, aniline blue, methylene blue, magenta, etc., are typical examples. Aniline black, perhaps the fastest colour known, is produced on the fibre by oxidation of aniline. Large quantities of aniline are used in this manner. It was from aniline that the first artificial colouring matter was discovered 1856, aniline purple or mauve. This discovery created a vast amount of attention, and as a consequence may workers came into the field and a large amount of research was carried out with aniline and

allied products, and the number of synthetic dyes gradually increased.

Aniline together with its allied products is a basis for the manufacture of a number of pharmaceutical products, which have been used as anti-pyretics, anti-febrin being one of the earliest though little used to-day. A later and well known one which is in a manner a derivative of an aniline compound is anti-pyrine.

Toluol is also used in a similar manner to aniline, but today is in more demand for the manufacture of an explosive called 'Trinitrotoluol' made by direct combination of toluol and nitric acid. From toluol an interesting medicinal compound 'saccharine' was synthesised. Saccharine is used in cases of illness, where sugar is not allowed, as in Diabetes, and is in a pure state 500 times sweeter than cane sugar.

Carbolic acid and cresylic acids find only a limited use in the colour industry, but are used extensively for antiseptic and medicinal purposes, a large body of soluble disinfectants rely on cresylic acid for their chief ingredient.

A numerous class of antiseptics, anti-rheumatics and anti-pyretics, such as Asceptol, Aristol, Salicylic acid and its derivatives; Aspirin, Salol and Phenacetin are derivatives of carbolic acid and its compounds.

Carbolic acid has a greater value in the manufacture of explosives, as it is fairly easily converted into picric acid by direct combination with nitric acid. The use of picric acid as an explosive is well known.

Naphthalene, a beautiful white crystalline body, is worked up into a great number of intermediate products of complex nature, such as Naphthalene-Saulphonic Acids, Naphthylamine, Naphthols, etc., etc., from which a large series of dyes are made. One of the most interesting of late years is artificial indigo, made by using naphthalene as a starting point. Artificial indigo has almost replaced the natural grown indigo. The manufacture of this dye illustrates the vast amount of time and money given to research by the German chemists. The carrying out of the research to a successful issue occupied twenty years, and cost the Badische Analine and Soda Fabrik upwards of $5,000,000. The only value of naphthalene in this country, up to the present, has

been limited to its uses as a deodorizer. The last of the series 'Anthracene' forms the bases of the 'Alizarine Colours,' of which 'Alizarine' is the most widely known and perhaps the most used. Alizarine when dyed yields a very fast red known as 'Turkey Red.' It was discovered simultaneously in England and Germany about 1869; before this time all the alizarine was obtained from the root of the madder plant.

The question of whether the artificial dyestuff and pharmaceutical industries could be established in this country has been much discussed during the past months, but considering the small consumption of dyestuffs and pharmaceutical products in Canada, it would seem hardly tempting enough to warrant one or more manufacturers spending a large amount of capital and time on the building of such a small industry, unless there was a possibility of it growing rapidly, or unless a market could be found in the United States or Great Britain. Taking the figures given by the Department of Trade and Commerce, the total imports of dyestuffs, including indigo, imported from Germany for the year ending March 31st, 1914, amounted to a sum of $244,259.00, and the value of the medicinal chemicals and pharmaceutical preparations during the same period amounted to only $59,512.00.

There is no doubt that such an industry could not be built up in a short time and give immediate relief. Much research and experimental work, requiring a vast amount of time and capital, would have to be carried out even if licenses were taken out to work a number of the German patents. The industry must be built up in such a manner that it would be able after the war is over to stand on its own feet and successfully resist the German competition which is sure to follow, and which is a competition backed by enormous capital, and capable of giving to the dyers not just a few dyes but dyes of almost every shade. That there would be a plentiful supply of coal tar by-products is beyond question. Although they are not insolated from the tar at the present time, if a demand arose they would be very soon put in the market. The following figures show that there is more than enough tar available in Canada to give all the by-products necessary.

The total number of coke ovens in operation in Canada at the commencement of 1914 was 1720, while 1375 were idle at that time. Of these coke ovens, only 730 are of the by-product recovery type, the remainder, being the bee-hive type, which type consume their own products in the course of operation. The amount of coke made in Canada during 1913 was roughly 1,500,000 tons. The consumption of coke in Canada is much greater than the production; thus the net imports in 1913 were roughly 650,000, making a total of 2,150,000 tons. If all this coke were made in by-product ovens in Canada a quantity of tar (equal to 20,000,000 gallons of tar) would be produced per year. As it is, the production is, roughly, under normal conditions about 9,000,000 gallons per annum.

The greater cost of installing the by-product ovens and the lack of markets for the by-products has no doubt hindered their substitution for the non-recovery oven. At the present time in Canada no benzol is produced from the coke oven gas. The recovery of this benzol is simply a question of demand, and at a price that would pay for the extraction, purification, and loss of heat value caused thereby in the gases. Supposing all the benzol were extracted from all the gases produced at the by-product ovens in existence in Canada to-day, there would result a quantity equal to 1,200,000 gallons—more than enough to swamp the country.

Closely connected with the recovery of benzol is another problem of great interest at the present time, on account of the big demand for picric acid for explosives and pure carbolic acid for medicinal and antiseptic purposes, and that is the synthetic production of carbolic acid from benzol. The process has been worked in Germany and it is now an open question whether this cannot now be undertaken here with profit.

THE CAUSE, PREVENTION, AND METHOD OF DEALING WITH UNDERGROUND FIRES

By S. A. Jones, District Inspector of Mines, Edson, Alta.

To be read at the Rocky Mountain Branch, August Meeting.

The purpose of this paper is to direct attention to the various causes of underground fires, to suggest the best methods to adopt in extinguishing them, and last, but what is most important of all, to indicate steps that may be taken with a view to their prevention. While it is the intention in this paper to refer to several of the causes of underground fires, the question of spontaneous combustion will take preference.

Fortunately, in this Western country, we have been particularly free from underground fires arising from spontaneous combustion; but with the development of our bituminous fields this danger is likely to become accelerated. In this latter respect, therefore, we have, in this Province, very little experience to guide us, and so it becomes necessary to take most of our illustrations from older countries, where many unfortunate experiences of this kind have taught valuable lessons on this particular subject. "Prevention is better than cure," and by profiting by experience in other localities, we may, to some extent, prepare ourselves for future eventualities.

Before, however, discussing at length the question of spontaneous combustion, reference may be made to other common causes of underground fires, chief of which are the following:—

COMMON CAUSES OF UNDERGROUND FIRES.

Use of Open Lights.—Open lights are most dangerous, particularly near stables, or where any inflammable material is kept, as the users of these lights are frequently very careless in throwing down burning wicks, which, of course, readily ignite the substances in their vicinity. As an example of this, may be cited the disaster that occurred at the Cherry Mine, Illinois, where 256 lives were lost simply because a torch set fire to a load of baled hay in a strong current of air at the foot of the down-

cast shaft. Also, in many mines the oiling or greasing of the mine-cars is done underground, and the black oil used for this purpose is often left uncovered, the oil-can being filled from the open barrel. In winter the oil has a tendency to freeze, and the writer has seen pails of the frozen oil warmed by the open torches used by the workmen. By this practice there is a likelihood of the oil being set on fire, with consequent dangerous results. On August 26th, 1908, in the Hailey, Oklahoma, No. 1 Mine, Haileyville, Oklahoma, a fire that caused the death of 29 men started in this way from an open barrel of oil.

Friction.—Friction is also another cause of fires. The heat so produced is most noticeable on brakes attached to wheels used at the top of self-acting inclines or on the brakes where a fast system of Main and Tail haulage is used. The friction produced by a fast-running endless rope on the rollers is sometimes sufficient to ignite the grease about them. Rollers have been known to catch fire in this manner at No. 3 Shaft of the Galt Mines, Lethbridge; but owing to the efficiency of the supervision of this mine, no serious result ensued therefrom. At the Great Western, a colliery in England, a fire caused by the sparks from a brake block igniting the brattice cloth resulted in the loss of 63 lives through suffocation by smoke from the fire.

Use of Electricity.—Fires may be produced by the use of electricity, in consequence of an inefficient installation, the use of cheap and inferior material, the use of unarmed cables, overloading of cables, incompetent workmanship in the erection of the apparatus, lack of cut-outs and resistance coils, and the accumulation of dirt, dust, paper, grease, and other refuse around the apparatus. Fires have also been caused by the short circuiting of the trolley wire, which may set fire to wooden frames of doors. Another cause of fire along entries is the grounding of power-lines on trolley-road returns. Power lines in mines are very seldom armored or even rubber-covered, and any movement of the timber may bring the bare-charged wire in contact with either wood or coal. If such lines are placed in roadways not often travelled, a fire may become serious before it is discovered.

In this connection, it might also be mentioned that the underground ventilating fan known as Booster Fans, and used to

help the surface fan, are liable to cause fire unless proper precautions are taken. These are being brought into use more and more, and they can be safely described as electrically driven. Usually, these fans are set in wooden frames, and owing to the proximity of this inflammable material, fires may be easily caused. In spite of daily inspection, there is likely to be a time when an armature gets over-heated and the insulating material takes fire. The remedy for this, and which may also prove economical in the end, is to have all mines, where electricity is used, governed by the "Rules for Installation and Use of Electricity in or about Mines," which will be found in the amendment to the Mines Act for the Province of Alberta.

Use of Furnaces Underground.—The chief danger from underground furnaces is the liability of firing the subjacent strata. There have been many accidents from this cause. In the Grassy Lake District, Alberta, a very primitive method of furnace ventilation was in operation. An iron bucket was used, and ashes accumulated, and these being hot set fire to the adjacent strata, thereby causing the abandonment of that particular opening. Even in mines where very elaborate furnaces have been constructed, the strata have been ignited, either through the excessive heat or through the conducting of the heat by the firing material of the furnace. One of these fires occurred at the Hamstead Colliery, South Staffordshire, England, and it was while dealing with it that Mr. Hughes, the Under-Manager, lost his life. In West Virginia, U. S. A., in December 1905, seven lives were lost in a mine on account of the timbers near the ventilating furnace taking fire, and producing enough smoke and poisonous gases to cause death. Once a fire of this character has started it is very difficult to deal with. The air space between the coal and the wall should be kept free from obstruction and all ashes removed from the mine.

Shot Firing.—Underground fires may also be caused when a shot has been fired and has ignited a small pocket of gas pent up in the coal. These ignitions will speedily set fire to the coal that has been loosened by the shot. Such an occurrence as this took place in the South Staffordshire Mines, England, in 1910, and but for the overman it might have had disastrous results. Blasting is probably the most fruitful source of mine fires, largely

because of the use of black powder, as a shot overcharged with black powder or dynamite is likely to set almost any bituminous coal on fire. The use of permissible explosives and inspection of the shots as soon as possible after firing tend to lessen the chance of fire from this source.

Spontaneous Combustion.—As, however, already stated, the most important cause of underground fires is spontaneous combustion. There are three main theories as to the cause of this:—

1.—Richter's. Absorption of Oxygen.
2.—Haldane's. Presence of Iron Pyrites.
3.—Pressure.

By experiment Richter proves that coal absorbs oxygen, and that one part of the oxygen combined with the carbon and hydrogen form carbonic acid and water, and the other part enters into combination with the coal and thereby increases its weight. He also found that large quantities of carbon-dioxide were given off before combustion actually takes place. In addition to this, he found that small coal absorbs oxygen more readily than large coal, and also that while the point of ignition for large coal is 370 degrees Centigrade, for small coal it is only 140 degrees Centigrade. As a result of this absorption of oxygen, it will be readily seen that heat takes place, and this heating continues until combustion actually occurs.

Haldane found that on the decomposition of iron pyrites, ferrous sulphate and ferric sulphate are formed. The action that takes place will be seen from the following equation:—

$$4\ FeS_2. + 15\ O_2. + 8\ H_2O. = 2\ Fe_2O_3. + 8\ H_2SO_4.$$

From this equation it is noticeable that the water promotes chemical action, and as at 38 degrees Centigrade oxygen is absorbed, local heat takes place in the vicinity of iron pyrites. One feature of iron pyrites is that the cuboidal form is not dangerous, whereas the radial form (Narcasite) is most dangerous. Richter found that iron pyrites of itself was incapable of commencing a gob fire, but nevertheless this local heating in conjunction with other action constitutes a real danger. It is a debated question how far the sulphur contents of the coal (visible as iron pyrites) is responsible for spontaneous fires. Formerly

it was believed to be the chief factor, but now the general opinion is that it is only an auxiliary factor.

Pressure is meant when the roof causes a general crushing of the coal, and during this crushing friction takes place, and this in turn produces heat. The pressure is most noticeable when the second weight comes on in long-wall workings. It has been found that this increase of pressure has the effect of increasing the temperature of the rider coal if there is one present. In addition to this heat being produced, this pressure is the means of forming microscopic cracks in the coal and these are avenues for air. In consequence of this it will be readily seen that the ratio of surface area exposed to the atmosphere is increased, viz.:—

$$\frac{\text{Ratio of surface area exposed to the atmosphere.}}{\text{Weight of the Coal.}}$$

Therefore, pressure not only produces heat of itself, but is the means of assisting in a large measure the process of oxidation referred to.

In the beginning of a gob fire, the circulation of air is insufficient to carry off the heat developed in the gob, yet enough air is present to cause slow oxidation. At first, then, oxidation is slow; with a rise of temperature more gas is liberated, and oxidation becomes more rapid. These reactions continue until the temperature reaches a dangerous point. The outward manifestations of a spontaneous gob fire are first that beads of moisture appear on the roof and upper part of the timbers at the edge of the gob.

Origin.—At one time it was generally believed that only bituminous coal was liable to spontaneous combustion, but it has been found that nearly all coals are. It is a well known fact that some seams are more liable to it than others owing to their nature and composition; but a combination of circumstances entailing the action of the processes previously referred to may take place in a seam that cannot by any means be termed fiery, and yet a gob fire will be the result. It has been generally found that thick seams are more liable to fire than thin seams, owing to the settlement of the roof. Then again, the coal cracks more

in thick seams, and, therefore, it is safe to say that the risk increases with the thickness. A better illustration of this point cannot be given than the case of the Hamstead Colliery, England, where 398 fires broke out in one year, namely, 1896. Here the seam worked was the South Staffordshire 10-yard seam.

Another frequent case of liability to gob fires is to be found when the seam worked has a rider coal over it. By a rider coal is meant a coal that is separated from the main seam, being worked by a few inches or feet of rock as the case may be, and which cannot be profitably mined. After the coal has been worked, large areas of waste are formed, and upon breaking down, the rider coal becomes smashed into a powder, and from this the action takes place. In long-wall working it is sometimes possible to recover some of this rider coal from the waste, particularly if there happens to be a good roof to which a few props can be placed until the rider coal is recovered. In the Lethbridge District, however, where the Room and Pillar system is in operation, and also where such rider coal exists, it is impossible to recover any of the coal in this way.

Another cause to which gob fire can be traced is the improper working of coal, *e.g.*, the size of pillars that are left should be so regulated that they are not too small, because if this be the case, more settlement of the roof takes place, and consequently more small coal is produced. The size of pillars should be regulated by the nature of the coal, the nature of the roof, and also the depth of the seam, and it is only by carefully watching these points that such danger may be eliminated.

Another cause is the leaving of small coal in the mine. In some districts small coal is of no commercial value, and the result is that the small coal is allowed to remain in the mine. This practice cannot be too strongly condemned, as it has been found to have been the cause of many gob fires.

Prevention.—In the first place, all small coal should be sent out of the mine, whether it be of commercial value or not; and in most cases a strong current of air should be kept passing along the faces in order to keep the surface as cool as possible.

In seams that are specially liable to spontaneous combustion, the panel system of working is the best to adopt, but whatever method is adopted only small districts should be opened up.

In mines where the liability to spontaneous combustion is great, a good measure of prevention would be to adopt a rigid and careful examination every four hours whether the mine is working or not.

Another preventive measure is the one adopted in one of the mines in Leicestershire, England. The seam is worked on the long-wall retreating method, and the face is worked in two operations because the seam is thick, being 14 feet. In addition to the regular packs which support the top coal a long pack running parallel to the face is put in every 30 feet. This pack

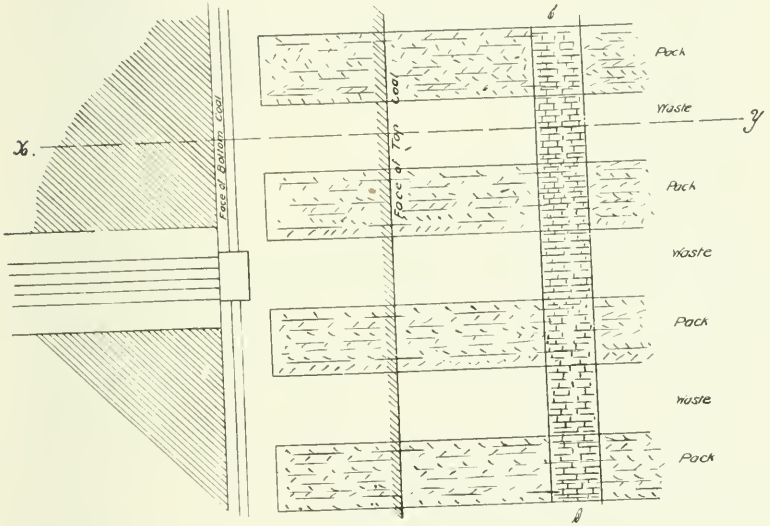
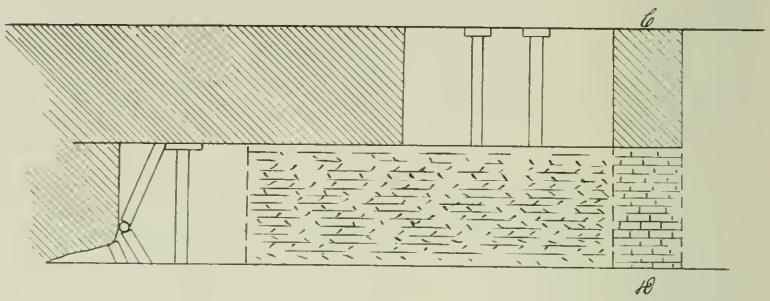


Fig. 1

is faced up with puddled clay and is known locally as a wax wall. This has been found to be a very efficient method, and as this mine has been practically free from fire for many years, it has proven itself to be very satisfactory. This method is fully illustrated and will be understood by consulting Figure 1, which gives full details.

A precautionary measure that should be adopted is the putting in of preparation stopping. This system is carried out in an excellent manner at the Mossfield Colliery, North Staffordshire, England, where a modified system of long-wall re-

treating is in operation. There stoppings are placed every 600 feet on the level. This seam is both fiery and gassy, and it may be of interest to state that it has been known to have all rails, gig-wheels, etc., removed and the two stoppings made complete in one and a half hours. This system will be easily understood by detailed Figure 2.



Section on X-Y.
Fig. 2

Method adopted after fire has broken out.—Mine-fires are usually small in area, and wherever practicable, the fire should always be dug out. To assist in the operation, water pipes should be laid to the outbreak and water used if necessary, although it is far better not to use water, as the steam in the atmosphere and its action on the roof may cause considerable inconvenience. This method of dealing with fires is always accompanied by a certain amount of danger to life from the smoke and gasses rather than from flames, but this danger has been considerably reduced of late years by the advent of the rescue apparatus.

A controllable ventilating system is of the greatest importance in fighting mine fires by active means; and the value of having a reversible fan is unquestionable. Numerous instances can be cited to show how both property and life could have been saved had it been possible to control the ventilating current.

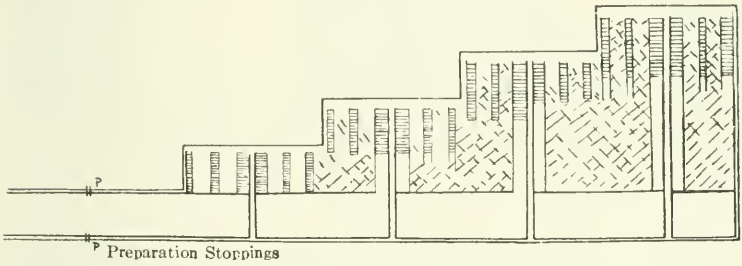
When the fires have attained such proportions that the means mentioned above fail, then more drastic measures must be adopted.

One method that can be carried out with a minimum of danger to the plant is the flooding of the mine with carbon

dioxide. An advantage of this is the fact that the ventilating plant at present in use is sufficient to convey the carbon dioxide into the workings. To produce this gas a large metal tank is suspended at the top of the downcast shaft, and this tank contains limestone or calcium carbonate. On to this calcium carbonate is poured hydrochloric acid, and the resultant carbon dioxide is formed as shown by the following equation:—

$$CaCO_3. + 2HCl = CO_2. + H_2O. + CaCl_2.$$

The ventilating fan is brought into use and forces this gas into the mine instead of pure air, and in turn puts out the fire. The one advantage of this method is that the mine can resume work after the fire is dealt with.



Sketch of one district showing how preparation stoppings are put in. Scale 50 ft. = 1 inch.

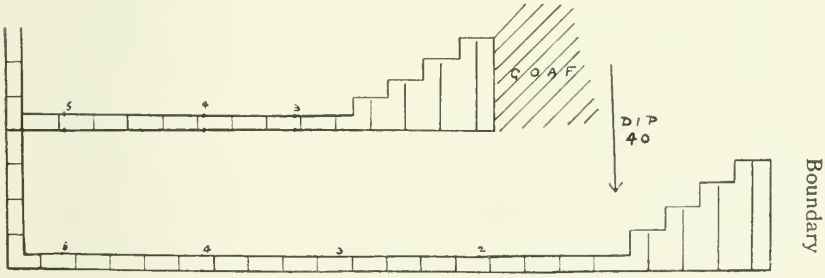


Fig. 3

Sketch of method worked showing where stoppings are placed in order as numbered, as coal is worked back.—Scale 200 ft. = 1 inch.

Another method of dealing with a gob fire is by means of flooding. It is obvious that this method entails great labour and trouble, particularly if the fire breaks out in highly pitching

seams. In some cases air chambers would be left where the water could not enter, and where smoldering could go on for any length of time, and where the fire would immediately break out again after the mine had been cleared of the water. The disadvantage of this method is the enormous damage that water does to the mine, especially where a fire-clay floor is found.

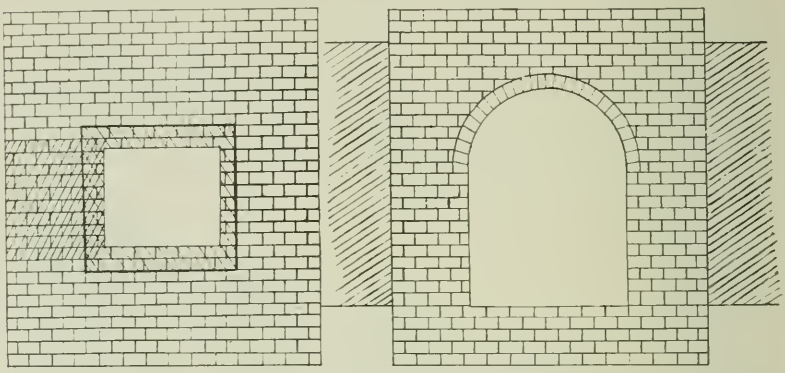
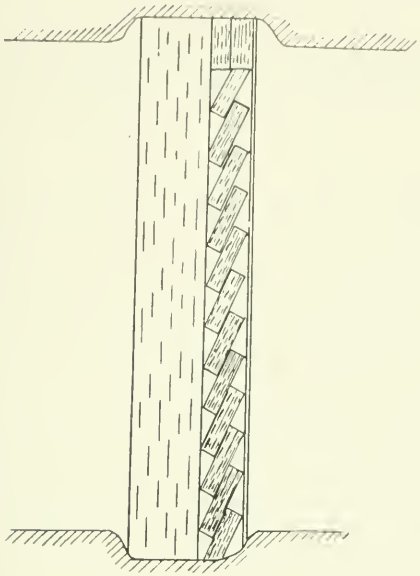


Fig. 4

As a last resort the fire may be dammed off. A good method of dealing with fire locally is fully illustrated in Figures 3 and 4. With the preparation stoppings a fire can be effectually sealed off with a minimum loss of coal area, and if it is necessary to take immediate action a dam can be made by nailing overlapping board on to the timbers and covering the whole with a covering of cement as is shown in Figure 5. It will be seen that such a dam will not resist internal explosions, and while this temporary dam is being built the ground should be prepared for a permanent brick dam. Great care should be exercised in the building of these permanent dams, the building of both the intake and return dams should proceed simultaneously up to nearing completion, not forgetting to place in each dam a 4″ air pipe fitted with a valve and pressure gauge. Whether the first dam should be erected on the intake or on the return side, or the dams placed on both simultaneously is a much discussed question, but in the opinion of the writer everything depends upon the circumstances and whether breathing apparatus and trained men are



Temporary Dam.
Fig. 5

available. When the dams are completed the air-valves should be closed. The pressure should be noted every two hours, and if found to be excessive a little of the pressure should be released, or the dam may be blown out. After a few weeks the pressure will, in all probability, decrease.

If the fire is an extraordinarily large one, it may be necessary to dam off the whole of the mine. In this case the fan is stopped, and a scaffold is fixed at a distance down the shafts. A good form of stopping in this case is to lay timbers on the lining of the shaft, and cover the whole with clay. This clay can then be flooded with water to the top of both downcast and upcast shafts. It will be seen that the great disadvantage of this method is the loss of the whole mine.

THE DRUMHELLER COALFIELD, ALBERTA.

By D. A. Macaulay.

The mines in operation in the Drumheller coalfields are situated in the valley of the Red Deer River, about 85 miles North East of Calgary, Alberta, along the Saskatoon to Calgary Branch of the Canadian Northern Railway. The approximate elevation of the valley is 2,300 feet above sea level. The valley ranges in width from three or four hundred yards to more than one mile. On both sides the ground rises with a steep slope to about 250 feet when it develops into rolling prairie land. The strata overlying the upper coal seams are exposed all along the valley and can be traced for miles without showing any sign of folding or faults. The measures lie almost horizontal. In places a slight inclination towards the west is noticeable. These measures belong to the Edmonton Series. The coal beds are generally assigned to the Laramie formation. While four different seams of coal are known to exist, as yet but two are being worked, namely, No. 2 (Newcastle) and No. 4. Both seams produce an excellent quality of domestic coal. as well as a good steam producing fuel. This is shown by tests made in the City Power House at Battleford, Sask. (Fig 1.)

The first workable seam in the series is No. 2, which has been operated for the past four years by the Newcastle Coal Co., Limited. The mine is located 1½ miles west of the village of Drumheller, and was opened by a tunnel driven in the coal under the steep slope of the valley on the south side of the river. The overlying border at this point is about 200 feet. The seam averages 4′4″ of clean coal, with the exception of one parting which is from one inch to four inches in thickness. The roof is comprised of hard green sandstone and requires but little timber to hold it. The floor consists of tough fireclay about 9 inches thick, which is removed in the entries to make a greater height. The mine is worked on the Double Entry System with Rooms and Pillars. Gas has not been found; nevertheless

Report of Cars of Coal Tested at City Power House, North Battleford, Sask.

Car No.	Rly. Wght.	Inv. Wght.	Our Wght.	Test commenced	Test completed	K.W.H output	Lbs. Coal per K.W.H	Cost per K.W.H	Calorific value	Per cent fixed carbon	Class of coal
49300 Twin City	60000	70600	71400	Ju. 10	Ju. 14	5145	13.7	3.25	9457	44.09	Run mine
310326 Yellow Head	60000	60000	57000	" 15	" 19	6480	8.8	3.27	13406	50.98	Run mine
663795 (Drumheller) Midland	60000	60000	64000	" 21	" 25	6435	9.3	2.74	. . .	. . .	Screen lump
54044 Twin City	72400	71600	.....	" 25	" 30	5362	13.3	3.15	9457	44.09	Run mine
33216 Cardiff	60000	61600	.....	Jly. 10	Jly. 15	4066	14.0	3.01	10286	47.30	Egg
304927 Pittsburg	59500	59500	67000	" 15	" 22	5856	10.2	4.36	.....	.....	Run dock
59398 Twin City	60000	74000	.....	" 22	" 27	5129	14.4	3.37	9457	44.09	Run Mine
311473 Jasper Park	61600	61600	.....	Aug.	Aug. 7	8745	7.5	2.79	15000	76.13	Run mine
37255	40000	36000	42000	Sep. 1	Sep. 3	4745	7.6	1.63	13000	. . .	Nut
(Drumheller) Newcastle 61716 Midland	60000	60000	.....	Oct. 4	Oct. 7	6892	8.7	1.87	....		Nut

Certified Correct,—

M. D. Cadwell,
Elect. Supt

Fig. 1

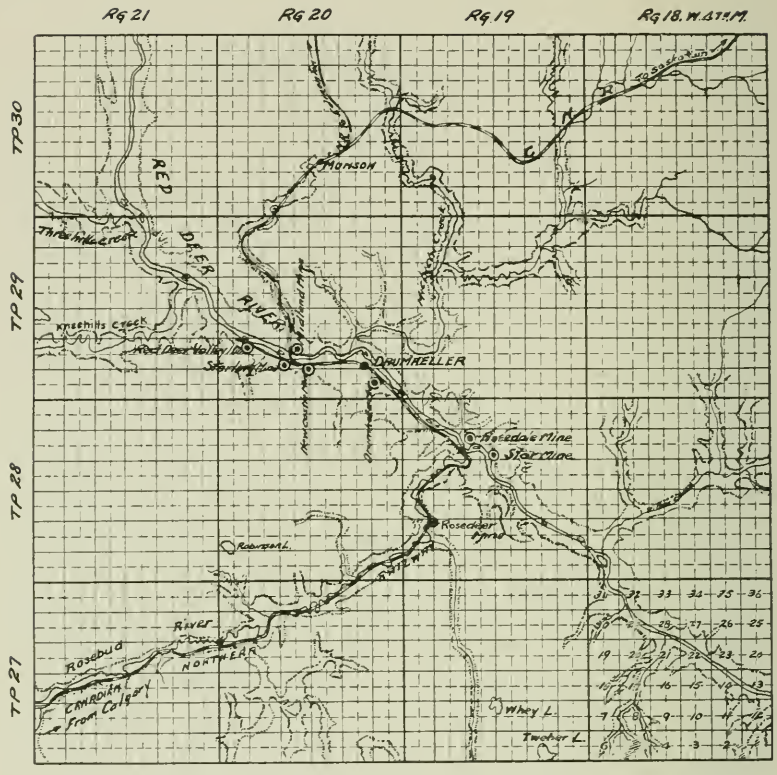


Fig. II.—Map, showing location of coal mines in Drumheller District, Alta. Scale, 3 miles = 1 inch.

safety lamps of the Kochler type are in use. The mining is done by hand picks, and loose black powder is used in blasting. The workings are above the river level and the mine is free from water. The maximum output per day is 500 tons.

No. 4 Seam, which is about 70 feet beneath No. 2, is operated by the Midland Collieries, Limited, the Drumheller Coal Co. Limited, the Stirling Coal Co., the Rosedale Coal and Clay Products Co., the Star Coal Co., the Red Deer Valley Coal Co., Limited, and the Rose Deer Coal Mining Co. Their location is shown on the accompanying map.

While in some cases these mines are separated by a number of miles, and considerable difference is shown in the sections of the seams, roof and floor, yet there is enough similarity to lead to the conclusion that all these mines are working on the same seam. A parting which is characteristic of the seam is found in all the mines mentioned. It averages about 15 inches thick and is composed of a band of grey clay from 3 to 6 inches thick, overlaying a very hard granular coal. Between this and the good coal of the bottom bench is a band of bone about three inches thick. The coal in the bottom bench varies in thickness at the different mines. In those where it is thin it would not pay to remove the parting in order to mine the bottom bench; consequently at several of the mines only the top bench is worked.

While the workings at any of the mines have not as yet covered a very large area, no faults or erosions similar to those found in the Lethbridge district, have been encountered. The roof is much better, and the floor does not show the same tendency to heave. These conditions coupled with the equally good quality of the coal produced enables the mines in this district to successfully compete in the market so long monopolized by the operators in the southern field.

The Midland mine is located opposite the Newcastle on the north side of the river. The seam was opened by a slope about 400 feet long, driven at a pitch of 19 deg. A three entry system was started from the foot of the slope. Rooms are broken off both back entries. They are 15 feet wide and 35 feet centre to centre. The rooms are driven about 200 feet and then cut off by parallel entries 10 feet wide. The roof and floor are both good and very little timber is required. The mine is ventilated by a 7′ 0″ Stine fan. During the first operations a large quantity of fire-damp was given off, but this has gradually decreased until at present no difficulty is found in keeping the mine clear. Safety lamps and permitted explosives are used, and all shots are fired by electricity.

The seam at this point is below the river level. At first about 500 gallons of water per minute had to be handled, now the quantity has decreased to about 35 gallons per minute.

Roof
Coal 2'-6"
Parting 1"
Coal 3'-4"
Bone 2"
Clay 4"0 6"
Gran. Coal 10"
Bone 3"
Coal 2'-2"
Bottom Bench Mining
Top Bench
Whole Seam Mined
Section of Seam at Red Deer Valley Mine
Roof
Coal
Clay 1½"
Coal
Parting
Coal
Bone 2
Clay 4"
Gran. Coal 8"0 12"
Bone
Coal 8"0 12"
Top Bench 6'-8"
Bottom Bench Not Mined
Section of Seam at Midland Mine
Roof
Coal 2'-11"
Parting
Coal 17"
Bone 1"
Clay 6"
Gran. Coal 6"
Bone 3"
Coal 4'-0"0 3'-6"
Whole Seam Mined
Section of Seam at Rosedale Mine
Roof
Coal 3'6"0 4'0"
Parting 1" Clay
Gran. Coal 1'5"
Bone 1"0 1½"
Clay 4'-2"6"
Bottom Bench Not Mined
Sand & Gravel 2'0 2'6" of Coal
Top Bench Average 6'-6"
Section of Seam at Star Mine
Roof
Coal 3'-0"
Clay 1"0 4"
Coal 6"0 8"
Bone 1"0 5"
Coal 2'0 3'0"
Whole Seam Mined Average Height 6'-6"
Section of Seam at Rose Deer mine

Fig. III.

Operations in this seam were commenced in September 1914, and the mine is now capable of producing 500 tons in two shifts.

At the Star mine, which is about 9 miles south-east of the Midland, the coal outcrops in the east bank of the river about 40 feet above its level. This mine was opened in June 1914 by two entries 55 feet apart, starting at the outcrop and following an easterly direction. Rooms are broken off both the main and back entry. Some of the rooms are 25 feet wide with a 13 foot pillar between; others are 35 feet wide with two tracks and two room necks driven narrow for 30 feet back from the entry. Only the top bench is worked. It is 5′ 6″ high, including about 3 inches of clay. Mining is done in the bottom clay and in the coal immediately above it. The top has to be spragged to keep it from falling on the miner. When the sprags are taken out the coal can be knocked down without using powder. In the entries the coal is mined by shearing in the middle and shooting both sides. Eighteen inches of brushing is taken up in the entries to make more head room. The roof consists of tough grey clod. Three-piece timber-sets are used in the entries and short sets across the track in the rooms; props are set in the gob. The rooms are driven 200 feet, then the pillars are drawn back; so far, but one pillar has been drawn at this mine. They succeeded in getting all the available coal, and secured the majority of the props before the roof caved. The tipple is placed on the opposite side of the river, and is connected with the Canadian Northern Railway by a short spur. In order to convey the coal across the river an aereal tramway was installed. This consists of a 1¼″ wire cable, securely fastened at both ends. On this is a two wheel carrier which is hooked over knobs that are fixed at each end of the mine car. The car and carrier is then hauled across the river by a ½″ rope having its both ends fastened to the carrier and wound around the drum of a small engine. The car is dumped at the tipple in the same manner as the ordinary tramway bucket and is returned to the mine by simply reversing the engine. The capacity of the car is 1,500 lbs. About 200 tons can be handled per day.

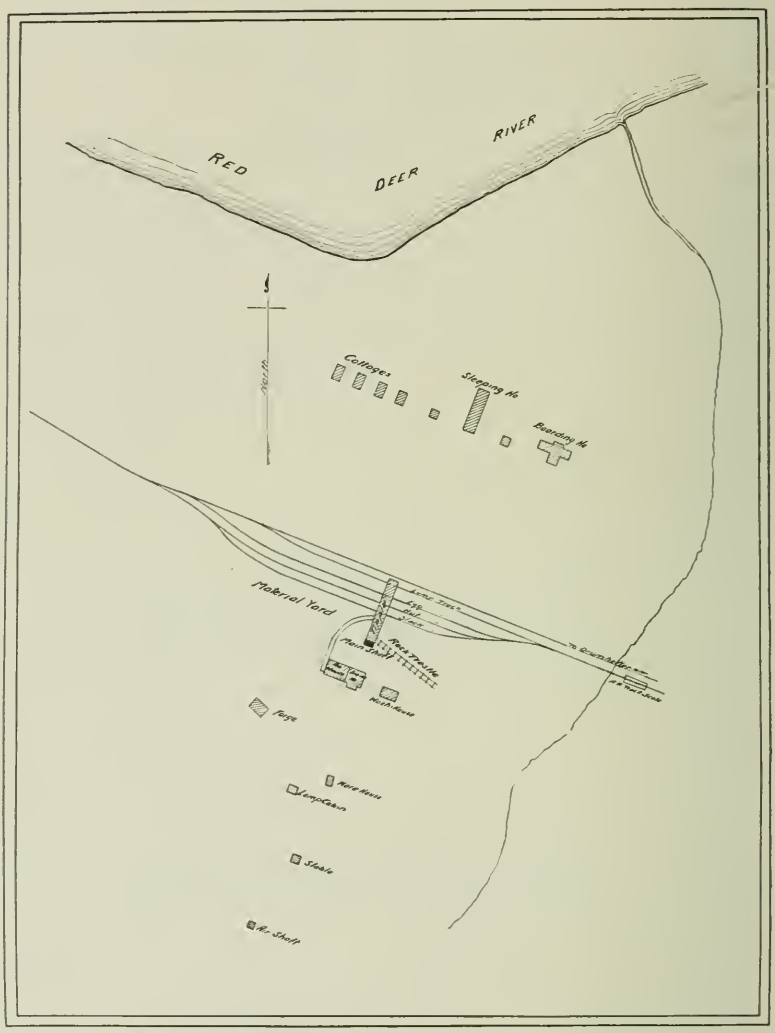


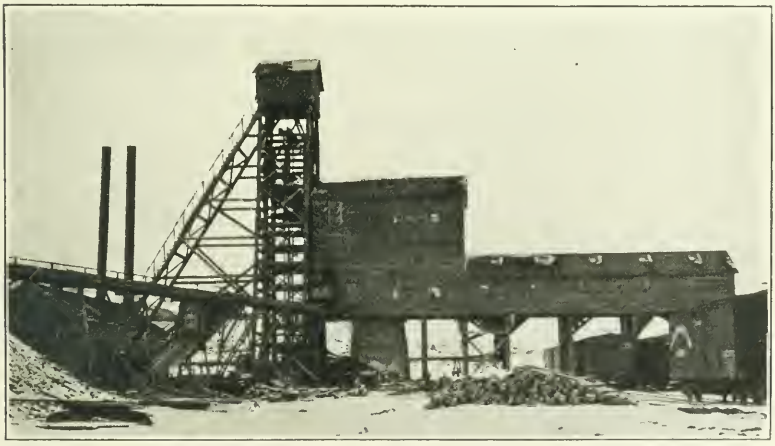
Fig IV.—Plan showing general lay-out of Surface Plant at the Red Deer Valley Colliery.

A mine equipped with modern machinery capable of handling 1,000 tons per day is operated by the Red Deer Valley Coal Company, Limited. The mine is located on Section 7, Township 29, Range 20, West of the 4th Mer. This company controls 3200 acres of coal lands. The mine is connected with

the Canadian Northern Railway by a spur 2 miles long Work on this plant was commenced in February 1913. A three compartment shaft, 180 feet deep, has been sunk to the No. 4 seam.

Surface Plant at Red Deer Valley Company's Colliery.

The workings have been laid out on the three entry system with double entries running parallel with the rooms. The whole seam is mined—its average thickness is 9′ 6″. A section of the seam is shown in Fig. 2.



The Tipple at the Red Deer Valley Company's Colliery.

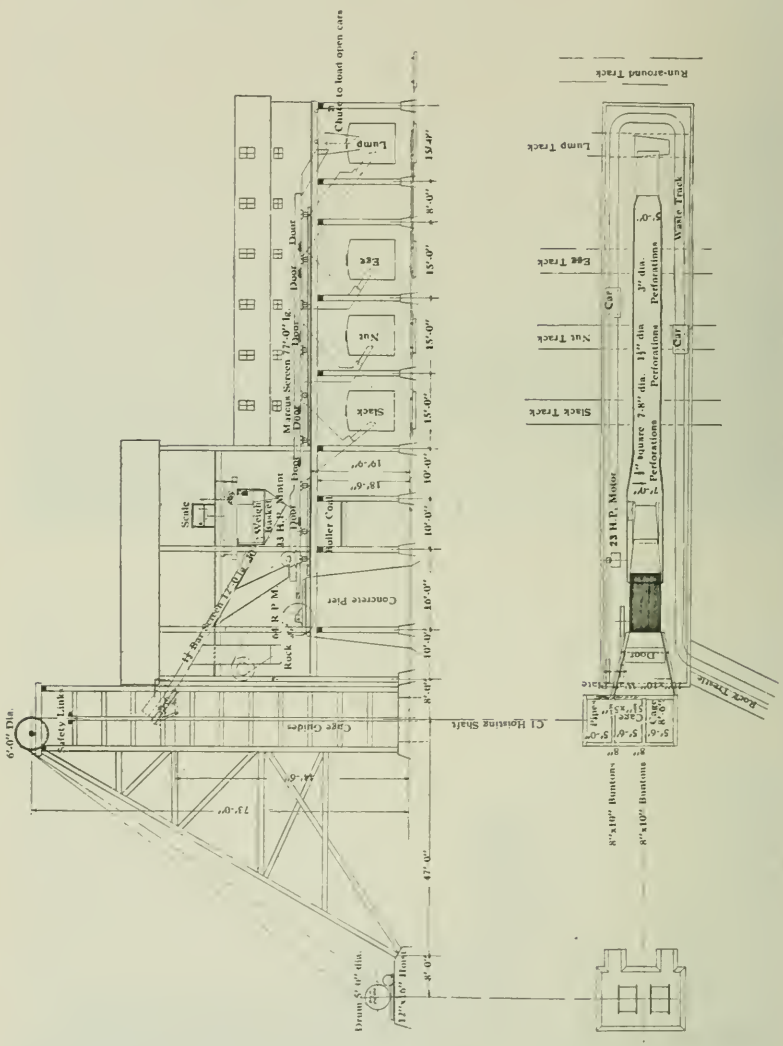


Fig. V.—Plan showing general arrangement of Tipple at the Red Deer Valley Company's Colliery, Drumheller, Alta.

At present the mining is done by hand picks, but preparations are being made to install mining machines. The general arrangement of the tipple and screening plant is shown in Fig. 3. The coal is hoisted on self dumping cages, working in balance. Before entering the weigh basket it runs over a $1\frac{1}{2}''$ bar screen.

Marcus Screening Plant at the Red Deer Valley Company's Colliery at Drumheller, Alta.

When ready to deliver the coal on to the "Marcus" screen the bottom of the weigh basket is allowed to drop by releasing a brake. After the basket has emptied itself the bottom is brought back into place by means of counter weights. A double deck Marcus screen, 77 feet long, having a capacity of 100 tons per hour, receives the small coal from the fixed bar screen and conveys it over the screen in advance of the lump coal from the weigh basket. This permits the small coal to be sifted through the perforated plates without being interfered with by the larger size. The perforated plates in the upper deck of the Marcus screen are bolted to "T" irons, and are easily removed thus enabling a change of classification to be rapidly made. Sliding doors are fitted in the lower deck and any mixture of the sizes produced may be loaded into the same car by simply replacing a door where desired.

The Marcus screen makes a good picking table as well as an ideal arrangement for screening coal. It is driven by a 23 H.P. three phase, 60 cycle, squirrel cage, alternating current

motor, with a full load capacity of 62 amperes at 220 volts. When the screen is running the motor takes but 35 amperes which is slightly one half load. The speed of the propulsion gear is 64 rev. per min., and the stroke is 12 inches.

The tipple building is heated by steam. Four lines of 1½" steam pipe are placed under the lower deck of the screen to prevent moist coal from freezing and sticking to the plates. A reducing valve is placed in the steam line which reduces the steam from boiler pressure to 25 lbs.

When designing this screen provision was made for loading lump coal into open cars as well as box cars. To the present the chutes for this purpose have not been installed, as the railway is now handling all the coal in box cars. These chutes are on the ground and can be set in place at any time desired.

The self dumping cage, shown in Fig. 4 was built by Herzler and Henninger Machine Works, Belleville, Ill. The car is held on the platform by cast steel horns which clamp over the thread of the wheels. To release the empty car at the bottom of the shaft knob "A" in the centre of the toggle joint comes in contact with a cast iron pedestal placed on timbers below the track and thus raises the counter weights to the position shown by the dotted lines. The empty is then bumped off by the loaded car going on the cage. When the cage is lifted the counter weights fall back into place bringing the horns over the wheels. When the cage approaches the dumping position lever "B" engages an angle iron fixed to the buntons, thus opening the catch just as the dumping wheels are entering the dumping brackets. One inch hoisting ropes are used. Load on rope; cage 6000 lbs., car 1500 lbs.; coal 3000 lbs.; total 10,500 lbs.

Eight-ton Ormerod's detaching hooks are installed in the headgear, beside the safety catches on the cage.

The power house is a wooden building containing the following equipment: One pair of 12"x16" self contained reversing engines with 2-5' 0" dia. 3' 6" face friction drums, built by the Vulcan Iron Works, of Wilkes-Barre, Pa. One 14" x 14" Robb-Armstrong engine, direct connected to a 125 K. V. A. 3 phase, 60 cycle, 240 volt A. C. Westinghouse generator, 277 R. P. M.

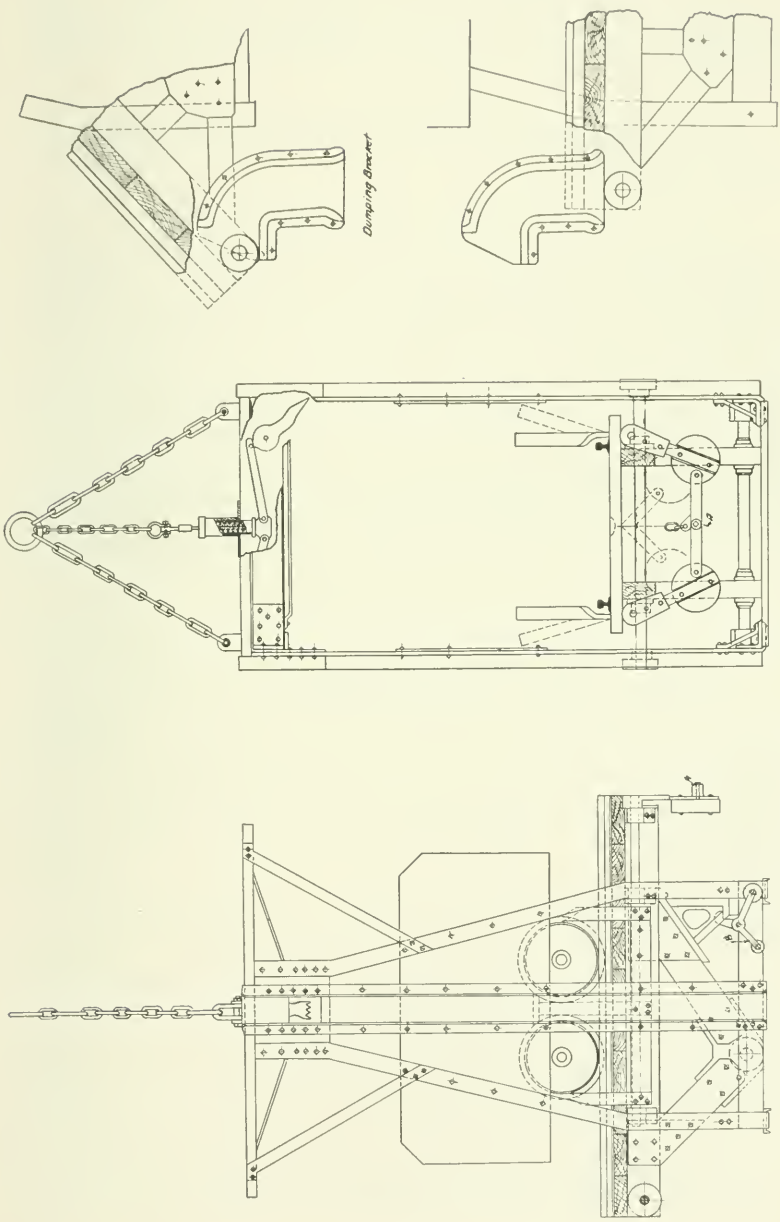


Fig. VI.—Self-Dumping Cage at Red Deer Valley Colliery.

One 6″ x 6″ Robb vertical enclosed type engine, direct connected to a 10 K.W. 125 volt, compound wound D.C. exciter, 400 R.P.M.

One three panel switch board.

One 10 K.W. 220 volt primary, 110 volt secondary, single phase transformer.

Two 18′ 0″ x 72″ Jenckes return tubular boilers.

The chief reason for installing a hoisting engine with two friction drums was that in case of an accident in connection with one of the cages, the other could still be operated. When the large generator is in operation the current supplying the lights passes through the transformer and is reduced to 110 volts; when idle, the lighting current is produced by the exciter. The mine is ventilated by a 6′ 6″ x 6′ 6″ double inlet Sirocco fan, capable of producing 100,000 C. F. M. against a resistance equal to 1½″ W.G. when operating at a speed of 190 R.P.M. The fan is belt driven from a 75 H.P. Westinghouse motor.

COAL ANALYSIS

Location	Moisture	Volatile Combustible	Fixed Carbon	Ash	Sulphur	Calorific Value in B.T.U.
Midland Mine, No. 4 Seam	12.98%	32.76%	49.74%	4.52%	0.62%	11120
Red Deer Valley, No. 4 Seam	11.55	31.82	46.75	9.88	.....	10440
Mouth of Rosebud River	*13.08	34.50	48.34	4.08	.....	9625
Galt coal	10.14	28.26	48.55	13.05	.....	.....
Taber coal, Upper bench	*11.36	26.64	45.60	16.40	.....	.....
Taber coal, Lower bench	*10.82	27.84	50.93	10.41	.....	.....

*Geological Survey, Coal Fields of Manitoba, etc., by D. B. Dowling.

NOTES ON MINE VENTILATION

By John Shanks.

Brazeau Collieries Limited, Nordegg, Alberta

Rocky Mountain Branch, November Meeting.

In this paper the author discusses a few of the principles of ventilation, which when opening up a mine should be considered. In Western Canada these principles have been frequently ignored with results not contributory to efficiency or economy; a consideration of the subject at the present time is moreover believed to be opportune in view of recent mine legislation in Alberta bearing on it.

The Alberta Mines Act was assented to on March 25th, 1913, and amended by Order-in-Council on Nov. 4th, 1914. The writer herewith offers the following criticisms and comments on some of the regulations of the act:—

General Regulation 16.—In every mine unless it is naturally wet throughout:—

(a) The cars shall be so constructed and maintained as to prevent as far as practicable *dangerous* coaldust escaping through the sides, ends or floors of the cars, but any cars which were in use in any mine at the time of the passing of these regulations may, notwithstanding that they are not so constructed, be continued to be in that mine until the 1st day of January, 1920.

(b) Systematic steps shall be taken to prevent as far as practicable explosions of coaldust occurring or being carried throughout the mine.

General Regulation 9.—After the 1st day of July 1915, there shall in every mine in which a mechanical contrivance for ventilation is used, be provided and maintained in a condition to be put into immediate operation, adequate means for reversing the air current.

General Regulation 16 aims at preventing as far as practicable the accumulation of *dangerous* coaldust. Does the word

dangerous refer to very fine or impalpable dust, or is it possible that the Government advisers are not yet converted to the idea that all coal dusts are dangerous under certain circumstances? The writer understands that the Government officials intend having the coaldusts from the different mines in the province tested with a view to their classification. Is this classification intended to make a discrimination between what is dangerous and what non-dangerous coaldusts? If so, the task is an extremely difficult one; and Rule 16 can only become operative after the results shall have been published. Since the results have been published of the laboratory experiments of Dr. Bedson in 1910-1911, and of the experimental gallery tests at Altofts and Eskmeals, in England, and elsewhere, mining men now have very little doubt concerning the explosibility of coal dust. Whatever difference of opinion there may be, it is generally conceded that the finer the coal dust, the more readily it gives off its volatile gases; and the lower the temperature at which it gives off these gases, the more dangerous the dust.

The following table gives the results of laboratory experiments with different dusts of equal fineness. The experiments were carried out some four years ago by Professor Bedson,[1] Armstrong College, Newcastle-upon-Tyne, England. The coaldust charges were projected, by a blast of compressed air, over a glowing platimun wire. The impulse communicated by the explosion was indicated by the extent to which a pendulum arrangement was deflected.

Coal used	Fineness	Weight	No. of amperes required to the platinum wire	Scale deflection centi-meters	Impulse
	Meshes	Milligr.			
Brown coal..	200	500	14	60.0	100
Bit. splint...	200	500	14	45.7	78.6
Lignite.....	200	500	14	41.5	71.0

[1] Trans. Inst. M. E., Vol. 39, part 5.

Owing to the large tonnage of lignite coal mined in this province, particular note should be taken of the impulse due to brown and lignite coaldust.

Coaldust on roadways is only dangerous because of the possibility of its being forced into suspension in the mine atmosphere by strong currents of air, caused by a local gas explosion; or due to a misjudged charge in a short hole; or, to a fall of roof over a large area. Authorities have computed that 10 lbs of coal ground into impabable dust, represents an absorbing surface of something between 10,000 and 60,000 sq. yards, and 3 ounces of coaldust thoroughly disseminated in 1 cu. yd. of air forms an explosive mixture. Judging from these figures one can see how difficult it is to overcome the coaldust danger.

The main sources of the coaldust found on roadways are:—

(1) Dust carried into the mines from the screening plant, and the tops of loaded mine cars in transit by the ventilating current.

(2) Leakage of dust from loaded cars during transit, a portion of which falls on the rails and gets ground into impalpable dust.

Making cars tight on the sides and bottom, while coal is allowed to be piled above the top of the cars, is only a prevention in a small degree. The coal above the top of a mine car is swept, as a rule, by the main intake current of air. This current carries away, in suspension, large quantities of fine coal dust; hence, the stronger the current and the finer the coal the greater will be the quantity of coal carried off in such a manner. As already stated, it is only when in a state of suspension that coal dust is dangerous; accumulations are only dangerous because of the possibility of their being forced into suspension. The author believes that more stress should be placed by engineers and mine managers to prevent, as far as possible, an unnecessary high velocity of the air current in the haulage roads. This can only be accomplished by increasing the area of haulage roads that are used as a main intakes; or, by maintaining a separate main intake, and ventilating the main haulage road as a split, carrying only sufficient air to supply the men and horses working on it.

The size of main intake and return airways requires careful consideration from the point of economy, apart from the question of safety which accompanies low-velocity air currents. Many properties have been hampered in their development by starting with unduly small airways. At the present time new properties are being opened, having airways far too small for the economical passage of the air necessary for future requirements. Of course, there is an economical limit to large airways, which is reached in thin seams. The following calculations will help to emphasize the writer's points. Suppose there are in a mine in which a five feet seam is worked two airways, X and Y. X has been driven ten feet wide in the five foot seam, and has not been brushed. Y has been driven twelve feet six inches wide in the coal, and three feet has been brushed out of the roof to make a height of eight feet. The pressure 'H' required to pass the same quantity of air in X and Y, other things being equal, varies as $\frac{O}{A3}$: where O represents perimeter of airways, and A area. Let H, O and A represent their respective value in airway X; and, H, O and A, similar values in airway Y; then, since 'H' varies as $\frac{O}{A3}$ we have:

$$\frac{H}{H_1} = \frac{O}{A3} = \frac{O_1}{A_1 3} \quad \frac{O A_1 3}{O_1 A 3} = \frac{30 \times 100^3}{41 \times 50^2} = 5.85$$

The pressure required for the small airway will therefore be 5.85, or, nearly 6 times as large as that required to pass the same quantity in the large airway, where the roof has been taken down.

Small airways mean: (a) a larger fan engine.
(b) greater coal consumption.
(c) dangerous velocities.
(d) high ventilating pressure and greater leakages.

From the above facts it is apparent that the question of required size of an airway is important, and should not be arrived at without consideration of all future requirements.

Sub-section (b), of General Rule 16, must be given the managers of this Province, matter for thought. If, the dust in the

mines which they manage is considered dangerous, they will have then to interpret what is exactly meant by "systematic steps." It has already been pointed out that it is almost impossible to clean up dust so thoroughly as to leave the roads in what might be called a safe condition. Watering of the roads must be practically continuous to keep the dust in a damp or caked condition, so as to prevent it from being forced into suspension through the medium of concussion or high velocities of the air currents. In winter time, owing to the water freezing in the mass, the caking process will be more effective. It appears to the author that the stone dust method (which requires, to be effective, equal portions of stone dust of equal fineness to that of the coal dust) must be very expensive in large mines. Such an authority on explosions as Mr. John Gerard (H. M. Inspector of Mines, Worsley, England) may be quoted. When taking part in a discussion as to whether preventive zones might meet the case, he said "Those who had seen the terrible force of the explosions at Altofts, must recognize that once an explosion of coaldust was initiated, the terrific force must involve a considerable loss of life before the preventive zone was reached. I have absolutely abandoned all hope of preventing loss of life by limiting zones. Coaldust explosions must be nipped in the bud."(1).

It will be interesting to have the views of the members of the Institute on the sub-section under discussion.

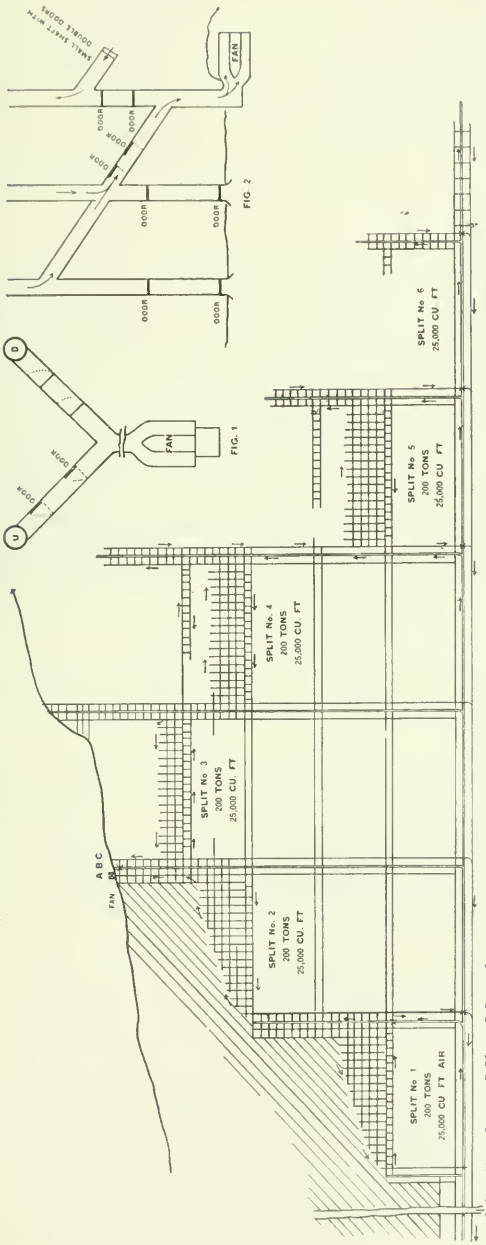
Rule 9 of the General Regulations makes it compulsory to provide means whereby the ventilating current can be reversed. A similar rule is in force in Great Britain, where a series of disastrous underground fires occurred, and caused the loss of lives which otherwise would have been saved if the air current could have been reversed. The reversal of the air current will be of more use in the case of underground fires in shafts, main airways, underground stables, etc., than in the event of an explosion, where the state of affairs underground is unascertained. It is a debatable question, whether, after an explosion, the fan should be restarted or not. Some authorities contend that it should be started only after preliminary exploration had been made by

[1] Trans. Inst. M. E., Vol. XXXIX, Part. V.

explorers wearing oxygen apparatus, who would ascertain conditions. To immediately reverse the air current after an explosion, without any preliminary examination of the workings, cannot be justified for a moment. The intake airway is usually the road to the surface; therefore, the workmen naturally proceed to reach this path, where, if the air current is following its usual course, they naturally expect to meet breathable air.

For the past few years, ventilating machines in Western Canada, and United States, have usually been so arranged that the air current can be reversed. This arrangement was originally made, not so much with a view to safety, as to economy. Because of the low temperatures prevailing in this country in winter time, great difficulty is experienced in operating wet shafts and haulage roads, owing to the rapid accumulation of ice. The heated air of the mine is frequently made to return along the haulage roads in order to keep the temperature above freezing point. To accomplish this it was found necessary to install a reversible fan. As already stated, it has been made compulsory for another purpose, to maintain means of reversing the air current in Great Britain. British engineers have adopted the plan of reversing the air current by means of drift and door arrangements. (See fig. 1 and 2,). The fan remains an exhaust fan (as is the usual practice there) at all times.

There is something to be said in favour of the exhaust fan, as against the force fan, especially in gassy mines having a large goaf area. With a force fan, the gases in the goaf area are always under a state of compression, due to the positive water gauge of the fan. If anything happens to the fan or engine to produce a sudden stoppage, the sudden release of pressure causes an expansion of the gases in the goaf, and they are likely to spread into the main arteries or airways of the mine, and cause serious trouble. The greater the goaf area, and the greater the positive ventilating pressure, the greater is the danger. Some authorities hold that the goaf area of a mine is about one-sixth of the space originally occupied by the coal. If this be so, then when a ventilating machine breaks down in a mine working a six foot seam which has a goaf area of one square mile, and bears an average positive ventilating pressure on the goaf of three inchesW .G., i.e. 15.6 lbs per square feet, 27,878,400 cubic feet of goaf gases



1½ miles from Mine Mouth.

expand inversely with the change in the absolute pressure. The calculation is as follows:—

	Lbs. per sq. ft.
Absolute pressure fan running 14.7 x 144 + 15.6	2,132.4
Absolute pressure fan standing 14.7 x 144	2,116.8

$$\frac{27{,}878{,}400 \times 2{,}132.4}{2{,}116.8} = 28{,}083{,}853 \text{ cubic feet.} = \text{new vol. of goaf gases.}$$

This causes a difference in volume of 205,453 cubic feet which can only find space by moving into the airways. This quantity of gas would charge, to its most explosive point, an airway 100 square feet in area and approximately four miles long.

Another danger arises from the general use of a forcing fan. Suppose a fire takes place in an airway, and, in order to keep the smoke and fumes from passing on to the workmen at the faces, it is found necessary to quickly reverse the air current, the total change in pressure will be approximately double the initial W. G. The change will be, say, from + 4 to − 4. This would cause a considerable expansion of the goaf gas, which in all probability would pass into the main airway and in this way would be carried in a cloud over the fire on its way to the fan.

In selecting a fan for a mine some engineers prefer to install a unit near the shaft or mine openings, capable of meeting all future requirements. In doing so, the equivalent orifice "A" of the mine at its maximum resistance, has to be assumed; then, a fan selected, whose orifice of passage "O" suits the proved principle, that, for high efficiencies, O should approximate to 3 A. Such a fan involves a large initial capital expenditure; and, in the early stages of development when the equivalent orifice of the mine is large, the efficiency of both fan and engine must be low.

Other engineers prefer to start with small units working near their capacity, and therefore full efficiency. The cost of foundation, etc., for motor driven fans units having capacities up to 75,000 cubic feet of air, on low water gauges, is not great; at least, not such as to make their installation and replacement by larger units unprofitable, provided a future use can be found for them when discarded in the first instance. The fans at the mines in charge of the writer are Jeffrey high speed fans, and are guaranteed to give 150,000 cubic feet of free-air against a pressure

of three inches W.G. The fans at present are belt driven by separate seventy-five horse power induction motors, capable of giving the fans a capacity of 100,000 cubic feet of free air against a $2\frac{1}{2}$ inch W.G. This would represent the fan and motor as giving a combined efficiency of about fifty-five per cent. At present the air passing in the mines is regulated down to 59,280 cubic feet; and the fan is working against a pressure of two inches water gauge. The meters show that the motor is taking eighty-five ampers at an average voltage of 435. This represents eighty-five horse power going into the motor for 18.68 horse power in the ventilating current: a very low efficiency caused by using a motor and fan too large for present requirements. These fans were installed in the spring of 1914. They are built so that they can be used as either forcing or exhausting machines. This coalfield has a long outcrop line, which rises in some parts to a considerable height above the mine mouths. It was originally intended to take advantage of openings in the high ground, and use the fans forcing in the winter months and exhausting in the summer months. By doing so, advantage would have been derived from the assistance of the natural motive column during most of the year. The writer found it impossible to use the fans forcing during the winter months, since the fan drift is also the water level. This drift would have choked with ice in a very short time.

The idea of reversing the air current for purposes of economy is worth considering in large coalfields in the mountains, where the outcrop is frequently reached at a considerable altitude, and where the total ventilating pressure is low. Of course, if the total ventilating pressure is high, the percentage of W. G. due to motive column will be low. In a coalfield situated as above, where the equivalent orifices of the mine splits are large, it appears to the writer that economical use can be made of small fan units, placed on the outcrop line.

Some such arrangement as is suggested in figure 1, seems practicable. The location of the outcrop fans would greatly depend on the configuration of the country, and the amount of overburden or glacial drift to be driven through. A fan of moderate capacity would have to be placed at the mine mouth, capable of ventilating the main levels and a few splits.

Suppose the plan represents the workings of a seven-foot seam.

Assume: Length of the main intake = 8,000 feet.
Area of the main intake — (12′ x 7′) = 84 sq. feet.
100,000 cubic feet per minute passes in main airways and ventilating splits 1—4—5 and 6.
4″ W.G. necessary to pass this quantity through the main intake and return alone.

If for any reason the outcrop fans at A x C have to be abandoned and the total mine ventilation, 150,000 cubic feet, has to pass through the main intake and return, then the W.G. required to pass this quantity through the main airways alone would be 9″. The ventilating unit at the mine entrance would have to be increased in power by two and one-quarter times. Two 60-inch diameter open running fans of the Ventura type, placed at A and C could each pass 25,000 cubic feet of air with a half inch W.G. This W.G. would be ample for such a split in a seven-foot seam. The horse power ventilating the mine, as suggested on plan, would equal 66.97.

The horse power in the air, passing the whole ventilation to a fan at mine entrance, would equal 212.92.

The difference in favour of the outcrop fans = 145.75 horse power in air. With a 60 per cent efficient unit this would be equal to 243 indicated Horse Power. Taking 2 lbs of coal per indicated Horse Power per hour, this represents a saving of 2,128.6 tons of coal per year; if valued at $2.00 per ton would equal $4,257.2.

The total cost of installing fan units locally would be approximately as follows:

Fan and motor for 100,000 cubic feet at 4″ W.G.	$3,300.00
2—60″ dia disc. fans 25,000 cubic feet each at ½″ W.G.	1,200.00
3 phase power line voltage 2,300 No. 8 wire, 8,000 feet	900.00
Total for combined installation	$5,400.00

The above figures show that large economies can be effected by the use of auxiliary fans.

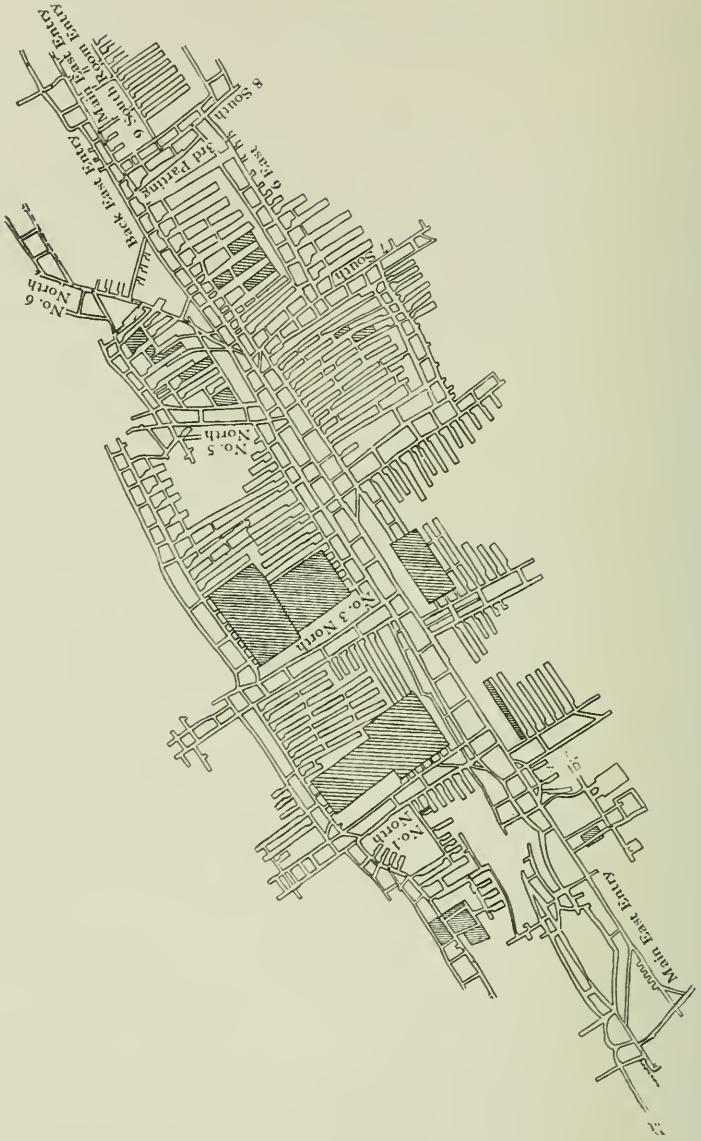
EXPLOSION AT THE TWIN CITY COAL MINE, EDMONTON

By J. D. De Hart, Edmonton, Alta.

To be read at Rocky Mt. Branch, August Meeting

On February 15th, 1915, a small explosion occurred at the "Twin City Coal Mine," whereby one man lost his life. An account of the explosion may be of some value as it brings out very clearly some points which are well worth noting.

The Twin City mine is operated by the Twin City Coal Company, Limited, and is situated in South Edmonton. The coal is a good grade of lignite and the seam, which is horizontal, varies from five to six feet in thickness. The mine is developed by means of two shafts: the hoisting shaft, which is rectangular and 167 feet deep, and the air shaft, which is circular and 181 feet deep. The roof is in most places a soft shale overlain by a fairly massive sandstone; but the thickness of the shale varies, and in places the shale runs out and the sandstone lies directly upon the coal. The mine is opened up from the shaft bottoms by two parallel main entries running approximately N. 51° E. These entries are known as "The Main East Entries." Off them cross entries are driven at right angles in both directions. These cross entries are known as Number 1, North; Number 2, North; Number 1, South; Number 2, South and so on. At the time of the explosion the last cross entries which had been driven were Number 8, South and the back entry to Number 8, South known as Number 9, South, and Number 6, North. Number 6, North is at an angle of about 40° with the main east entries and was connected to the back east entry by a room entry. From seven South a room entry had been driven east parallel to the main east entries. Off this room entry, which was known as the "Six East Entry," rooms had been driven, but the entry had been abandoned, although all the rooms were not finished nor any of the pillars drawn. A connection had since been made between the inbye end of six East and eight South, and the six inside rooms had been opened up. These rooms had been worked on the day of the explosion and were being cut for the next day when the explosion occurred. A room entry had been driven parallel to the main east entry and 100 feet from it off 9



Plan of Twin City Mine

South and this was known as the 9 South room entry. It was connected through to 8 South. There were nine rooms opened off this room entry. The first six had been driven up their full distance and the pillars were being drawn. Near the face of the

main east entry a crosscut had been started to meet the 9 South room entry. The face of this crosscut had advanced about 30 feet from the main east entry.

The mine is ventilated by a Sheldon fan which delivers about 24,000 cubic feet of air per minute when running at a speed of 150 revolutions per minute. This is the normal speed at which the fan is operated and at the time of the explosion the fan was forcing air through the mine. Nearly all the stoppings in the mine are built of boards, but a few temporary stoppings near the faces are of brattice cloth. Open lights are used in the mine; but as gas had been reported in the main east entry men working on the return from this point had safety lamps on the day of the accident.

The explosion occurred at about 5 p.m.; but curiously enough no word of it reached the surface or any of the officials until 7 p.m.

The afternoon shift went on at four o'clock. It consisted of a few miners who filled the empty cars, two drivers, two track cleaners, the timbermen and a fireboss. At the time of the explosion the men in the mine were supposed to be working in the following places: Number 1 North, six men; Number 3 North, three men; Number 6 north, five men; 6 East, two men; on the main east entry, timbering and track cleaning, eight men; and two drivers, who may have been anywhere between 6 North and the pit bottom or in third North or first North. It so happened that the fireboss, on going down at four o'clock, was detained at the stables owing to some trouble with the horses, and, instead of going inside at once as usual, decided to come up for his dinner before going around the mine. The fireboss was about to go down again at about 7 p.m. when one of the track cleaners came up to the surface and stated that someone had lit some gas in the main east entry. The fireboss went down at once taking the track cleaner in as far as the third parting, which is just outside of 8 South. At the third parting there was a good deal of smoke, and the fireboss left the track cleaner there with orders to go for help if he did not return, and made his way down 8 South and into 6 East where a machineman and his scraper were at work. The fireboss asked these men which of them had lit the gas, but they assured him that it was not they but that they had seen a light go into the main east just before

the explosion and that they thought the man whose light they had seen was dead. Up till this time the fireboss had seen no signs of any explosion and, probably because there had been nothing to show that there had been an explosion and not merely an ignition of gas, he left the men in 6 East, omitting to send them out, and made his way into the main east. He went to the face of the main east entry and crosscut, but the air was bad and very smoky and he saw no trace of anyone there. He then made his way out and started to make enquiries as to who was missing and had just ascertained that it was probably a man named Smidt, who should have been in third North when the mine manager met him. They sent a man to the surface for more safety lamps, and then went back into the face of the main east and into the face of the crosscut. Nearly all the timbers had been blown down and all the stoppings blown out from a point about 100 feet inside of 8 South to the last crosscut between the main and back east entries, and there were several empty cars which had been somewhat twisted by the force of the explosion. Where the timber had been blown out, from a foot to two feet of roof had fallen. This was the shale, and it left the sandstone forming a good roof. Owing to the fact that the main east entries are quite wet from 8 South to the face, and that the mine is not dusty, the effects of the explosion were limited to a radius of about two hundred feet. Both the manager and fireboss lost their lights owing to the smoky atmosphere and to the fact that they were both staggering somewhat from the effects of the gas. On getting into fresher air they immediately went into 6 North and collected the men there. They decided that it was too risky to send anyone for the two men in 6 East, and, as the effect of repeated doses of carbon monoxide is cumulative, both felt that they themselves could not possibly manage to get there. The men from 6 North were therefore taken along the entry and on the way the men working in 5 North, 3 North and 1 North were collected. All open lights were extinguished before these men were allowed onto the return although no signs of gas could be detected there. This left only three men in the mine, Smidt who had lit the gas and the two men in 6 East. These last two were on the return from the afterdamp.

One man had been sent ahead to go to the surface and take down the names and check numbers of all the men as they came up. As soon as the men reached the pit bottom they were hoisted as quickly as possible, and immediately the last man was out of the mine, the fan was reversed and driven as fast as possible. The engineer stayed at the fan except when he was called to the hoist and was able to keep it running at about 300 revolutions per minute.

It was known that there were two men in 6 East and that Smidt was still in the mine, but a careful comparison of the names of the men who had come up with the time book and the checking board showed that everyone else was out of the mine. Telephone calls were sent out for help and in a remarkably short time several of the managers from the nearby mines were on the spot. Unfortunately no rescue apparatus was available; but a party was at once organized and proceeded along the main east entry with brattice cloth, axes, nails, etc. Just inside of 7 South a brattice stopping was erected so as to force all the air down 7 South and along 6 East. The party then proceeded cautiously down 7 South and along 6 East. Some traces of afterdamp and smoke were encountered in 6 East, but the brattice stopping placed on the Main East entry was sending fresh air along the 6 East entry, and the party were able to make their way along until, at about 10 p.m., the machineman and scraper were discovered. These two men were still at work. They had been cutting a room which was not yet connected to the rooms on either side and so were in a dead end. The compressed air from the machine had prevented the afterdamp from going up the room. Two of the party took them to the surface and the rest proceeded to cautiously follow the afterdamp along 6 East and up 8 South. On reaching the 9 South room entry it was decided to explore this first as it was easier to clear the gas from there than from the Main East. With this end in view a brattice stopping was erected across 8 South, just north of the room entry, and the party proceeded a little way along the room entry. Here a good deal of smoke was encountered, but it was decided to try to get to the face of the room entry and also into the Main East entry. Both these places were therefore explored, but without any result, so that it be-

came almost certain that the missing man was either in one of the rooms or, as was more probable, under one of the caves in the Main East entry. In either case it was impossible that he could be alive owing to the afterdamp, and, the air being bad, the party returned to the surface. On the way out they met Mr. Stirling, the Chief Inspector of Mines for the Province. Henceforward Mr. Stirling took charge of the work. Another party was organized without delay and started inside. As the two men who had been in 6 East were now out of the mine, there was no need of the brattice which had been put across the Main East entry at 7 South, and this was now torn down. The party proceeded along the Main East entry to 8 South and removed the brattice stopping placed there by the former party. By this time the air had cleared a good deal and the rooms and pillars of the 9 South room entry were all explored but without result. A start was then made on the Main East entry and, at about 4 a.m. on February 16th, the body of Smidt was found under the second cave. The party brought the body to the surface and all were out of the mine at about 5 a.m.

The explosion illustrates many points that are well worth noting. Most curious is the fact that the accident occurred at about 5 p.m. while nothing was known about it by the officials until 7 p.m. At the time when the explosion occurred the machineman and scraper who worked in 6 East were starting down 8 South with their machine. They saw a light going east on the Main East entry and shouted. Undoubtedly this was Smidt's light and he answered. About a minute later the explosion took place. Both the machineman and his scraper were knocked down and their lights extinguished. Nevertheless after about five minutes they rose, lit their lamps and went to work. When the fireboss reached them they assured him that the man whose light they had seen was dead, but in spite of this they were still at work when they were found by the rescue party at 10 p.m. The machineman in 6 North had some trouble with the compressed air and visited the Main East to try and discover what was wrong. He saw signs of an explosion, but in spite of this, he likewise went quietly back to work. Apparently all the men working on the Main East entry felt the explosion; nevertheless they also continued to work. The miners in 6

North debated whether it was an explosion or a shot. They decided that it was a shot and continued working. This is, perhaps, typical of the class of labour with which the mines in the Edmonton district is inflicted. A certain percentage, of course, represents good miners, but a very large number are foreigners with little or no knowledge of mining.

It may be said generally that in the case of an ignition of gas, it is wise to withdraw all the men from the mine. Had the fireboss sent out the men in the 6 East when he first reached them, a good deal of trouble and anxiety would have been saved.

Another point which is shown very clearly is the fact that a reversible fan may, at times, be the means of saving lives, but it should not be reversed without thought as to the results. Had the fan been reversed before the men who could be got at were brought out of the mine, the lives of all these men would have been endangered. As it was, the reversal of the fan probably saved the lives of the two men working in 6 East.

The explosion shows that the men should never be allowed to leave their working places except in cases of danger. The man who was killed had been working in the crosscut off the Main East entry on the Friday. Apparently he had left something there, and on the Monday he left his working place about 1500 feet away and walked right into the gas with an open light. The men had always been kept to their own places very strictly. It is clear, however, that it is impossible to always prevent a man leaving his working place, and for that reason mixed lights can never be safe. Where there is dangerous gas in one part of a mine, nothing but locked safety lamps should be allowed in the mine.

The experience of the manager and fireboss who lost their lights in the after-damp and got out with difficulty, shows that it is a very wise precaution when going into after-damp to take an electric lamp.

Great credit is due to the mine managers from the neighbouring mines who so promptly responded to the call for assistance, and who were so ready to go down a strange mine and do all that was possible to rescue those left in and to recover the body of the man who was killed.

RADIUM

By F. A. Ross, Spokane, Washington

Western Branch, 1914

"'The Wonders of Radium,' both fact and fable," have been treated so fully in scientific and popular literature that this article will be confined to the commercial phase of the question rather than to its purely scientific side, although the latter will be discussed briefly to give a clear understanding of the medicinal (and, therefore, commercial), value of radium salts.

For the purpose of this discussion, the subject has been divided into the following four principal heads:—

1. Radioactivity and the characteristics of Vanadium, Uranium and Radium.

2. Radium-bearing ores and their occurrence.

3. Governmental action in respect to radium-ores and the reasons therefor.

4. The business and mining propositions involved.

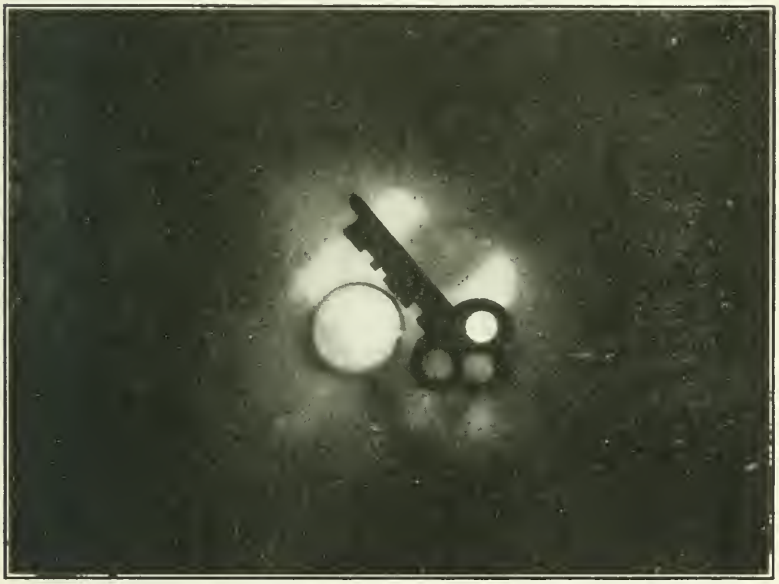
A general acknowledgment is due to sundry scientific and technical publications for the data herein quoted and to several well known specialists as well, the total number of authorities being too large to permit of specification at this time.

Radioactivity and Emanations.

Radioactivity has been defined as the manifestation of a disintegration of the atom—a process that is spontaneous and constant; emanation is defined as the disintegration substance itself.

It is well to consider for a moment the tremendous advancement of science as indicated by this rejection of the long-held theory that the atom is the ultimate division of matter:—it is a disintegration of this *atom* that produces the phenomenon called radioactivity. At the beginning of the

Christian era there were but seven known elements, copper silver, gold, tin, lead, iron, and mercury. In the 1750 years following this period only ten more elements were discovered, making seventeen in all. But in the next 140 years, fifty new elements were added to the list, totaling sixty-seven in 1891, which total was increased to eighty-two in 1912. The 18th Century, then, marked the beginning of that astonishing scientific advancement that made the 19th Century an epoch



Steel Key and Ring on Pitchblende Ore.

of human achievement and which prepared men for any surprise that the 20th Century may afford.

Returning to the subject of radioactivity, modern methods of analysis and the latest instruments of precision show that very many substances, including natural and artificially prepared waters, are distinctly radioactive. Potassium, for instance, has a specific radioactivity, emitting Beta rays that equal those of uranium except that they are of less electrical effect. Moreover, it is stated that radium itself is present

almost everywhere, although in infinitesimal quantity except where concentrated in the form of ore.

Three kinds of emanations, or disintegration particles, have been distinguished and named the "Alpha," "Beta," and "Gamma" rays. From radium salts, these rays are given off in the proportion of 90% of Alpha, 9% of Beta and 1% of Gamma rays.

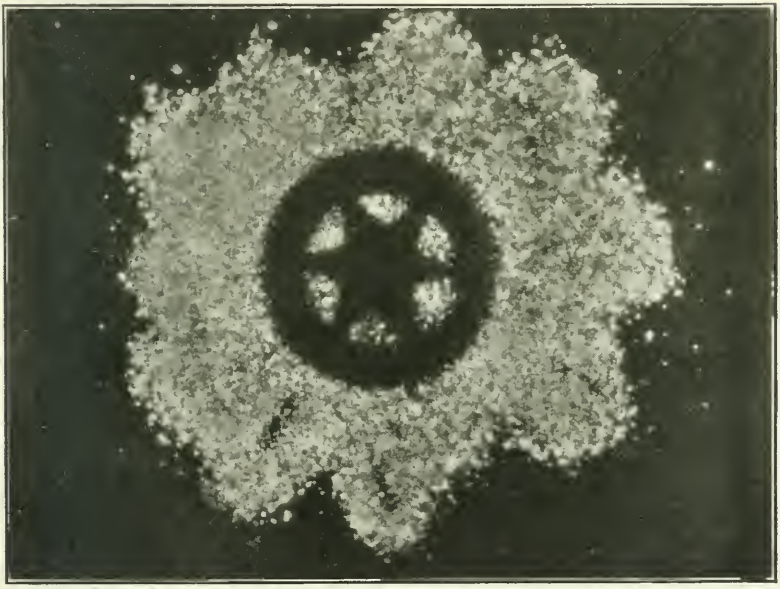
The emanation substance, itself, is a material of greatest interest to the physicist; it appears in the form of a gas that

Pitchblende Concentrate on Lead-Stencil. 30%, 54 hours.

may be isolated and weighed; it is soluble in water and is capable of passing through many substances. Also, it constantly develops energy in the form of heat, which calculation shows to be equal to 20,000 horsepower per hour from an equivalent of 2.2 pounds of emanation. Again, this emanation may be condensed to a liquid at about – 150 degrees, Centigrade. Glass and gems are slightly coloured by it; water is slowly decomposed into hydrogen and oxygen and photographic dry-plates are darkened; if strongly concentrated it is observed

to glow in the dark and to render other substances luminescent; also, it ionizes the air which then conducts electricity.

In short, this emanation is, itself, an unstable substance that, in successive disintegrations, produces first, the Alpha, then the Beta, then Gamma rays and, finally, helium which appears to be the lowest step of disintegration.



30% Pitchblende Concentrate. 49 hours on aluminum.

DISCOVERY OF RADIUM

Apparently we owe to pure accident the discovery of that radioactivity in uranium ores and salts that, in 1896, started Madame Curie upon the now famous quest that resulted, two years later, in the isolation and announcement of radium—a discovery that has advanced physics by leaps and bounds, and which has brought much nearer to us that sole guarantor of all terrestrial life, the sun—93,000,000 miles away.

OCCURRENCE OF RADIUM

So far, radium has not been found in commercial quantity dissociated from uranium or uranium minerals and these, in turn, are often associated with vanadium; therefore, a brief notice of these metals may prove interesting and instructive.

Vanadium, discovered in 1830, is highly valued as an alloy in iron, steel, copper and bronze. In suitable proportions it acts as a scavenger in removing harmful nitride and oxide

A—Concentrate; *B*—Pitchblende Ore; *C*—Lead Letter; 72 hours.

gases; it unites with carbonless metal and toughens it; also, it acts upon included carbides in a manner to strengthen and distribute them, preventing segregation in the ingot and increasing the elastic limit and tensile and torsional strength.

Uranium, discovered by Klaproth in 1789, is scientifically remarkable in many ways, but commercially its principal use is in the production of velvety-black glazes on pottery and a peculiar greenish-yellow iridescence in glass which then becomes radioactive, often strongly so.

Uranium is estimated to have a life of about 10,000,000,000 years before it becomes transformed, by disintegration, into uranium X, with a life of twenty-two days; Uranium X transforms into ionium, the life of which is not yet determined; and ionium transforms into radium with an estimated life of over 2,000 years. Radium, itself, disintegrates into radium emanation and then successively down through a series known as Radium A, B, C, D, E, and F, the final and ultimate stable form of this wonderful metal being ordinary lead, according to the present supposition. Seven of these disintegration products give off Alpha rays, three give off Beta rays, Uranium X gives off Beta and Gamma rays and Radium C gives off all three rays—Alpha, Beta and Gamma.

Thorium, discovered by Berzelius in 1828, is a most important radio-element, very valuable in the manufacture of Welsbach mantles for incandescent gas lights which are impregnated with salts containing 99% of thoria and 1% of ceria. Its principal ore is monazite, found in abundance in Utah, North Carolina, and in the placer sands of Idaho.

In fact, the known rare elements may be divided into two general groups; namely the Uranium Series and the Thorium Series, but it is the Uranium Series only that is considered in this discussion, the medicinal value of thorium disintegration not having been fully determined as yet.

Radium Ores

The commercial ores of radium, which, at present, means the ores of uranium, are principally those containing carnotite and pitchblende, or uraninite, a lead-uranium oxide. Of course there are other radium-bearing minerals, some fifty, in fact, composed of uranium in combination with other oxides and phosphates, such for instance as autunite, a lime-uranium phosphate, and torbernite, a copper-uranium oxide, both of which occur in important quantities in other countries, but always they are subordinate, commercially, to carnotite and pitchblende.

Carnotite.—Carnotite, said to have been named in honour of President Carnot of France, is a canary, or lemon-yellow,

mineral occurring in sedimentary formations, usually in pockets in sandstone or as incrustations upon the pebbles of conglomerates, and also in small seams or cracks in both. Its specific gravity is about 4 and in composition it is a triple oxide of uranium, vanadium, and potassium combined with water, the formula being $K_2O.\ 2UO_3.\ V_2O_5.\ H_2O$. Sometimes it is disseminated through sandstone in the form of light yellow specks. When freed from its gangue it is a yellowish powder that is very irritating to the eyes and upper air passages as well as to the bronchial tract. Again, carnotite is sometimes found in a comparatively massive form associated with other blue, black, or brown vanadium ores. Because of this erratic occurrence and of its variation in colour, carnotite is a difficult mineral to follow; its mining, to date, has been confined to outcrops or along sections cut by streams, gulches and canyons.

By far the largest of all known deposits of radium-bearing ores are the carnotite beds of the Paradox Valley in San Miguel and Montrose Counties, Colorado, and in the adjacent regions of Utah—a recognized area of some 500,000 acres. These deposits have been known since 1881, when they were mined in small lots for their gold content and, later, for their vanadium and uranium. As yet, geologists are uncertain as to origin of these deposits, but a number of specialists are in the field and more accurate information may be expected shortly. Suffice it to say here that, inasmuch as vanadium is widely distributed in sandstones, limestones and igneous rocks, these deposits may be the result of solution and re-concentration under the three conditions most essential to all such ore deposits, namely, time, heat, and moisture—all of which have great influence in just such semi-arid regions as those of Colorado, Utah, and Nevada. The presence of uranium in these carnotite beds is more difficult to explain since the nearest pitchblende yet found occurs in Gilpin County, several hundreds of miles away, but time and closer study may offer a solution to this problem also. The sandstones of this region have been found to possess a much higher radioactivity than those from any other known locality. South Australia is the only other country in which carnotite has been found in quantity, and here it is associated mainly with titanic iron (ilmenite),

which fact may well serve as a pointer in prospecting in the United States and Canada. In Colorado and Utah, the carnotites are associated with other vanadium ores and with gypsum, vegetable fossils, carbonaceous matter and, often, with the carbonates of copper. The colour of carnotite is not always a lemon-yellow, it is often a grayish-brown or black, although the yellow colour follows weathering or heating.

There is great need of an efficient system of concentration, by wet or dry methods, in the case of all low-grade radium-bearing ores, since the aggregate loss in present methods is very great.

Pitchblende.—While carnotite is a triple oxide of potassium, vanadium, and uranium and is found in sedimentary formations, pitchblende, on the other hand, is usually an oxide of uranium combined with varying proportions of lead, copper, iron, bismuth, manganese, lime, silica, and aluminum, and it occurs chiefly in granites, gneisses, schists, and igneous rocks, associated with the precious metals and sulphides of the baser metals.

Its lustre is submetallic, dull; fracture, conchoidal; brittle; colour, black to dark brown, olive-green, grayish. Specific gravity, about 9.7; hardness, 5.5.

It has been found in Connecticut, North Carolina, Texas, South Dakota, Quebec, Saxony, Austria-Hungary, Norway, Spain, and East Africa, but the largest of the known pitchblende-bearing veins lie in Gilpin County, Colorado, where they occur in gneiss and mica schist in more or less close proximity to intrusive andesite and granite. The ores of these veins were originally worked for their gold and silver and were usually free milling in the oxidized zones, but sulphides appeared with depth and, in most instances, when pitchblende appeared the gold disappeared and the mines were abandoned, but with the new demand for pitchblende they are again being developed.

Unconfirmed reports of pitchblende occurrences have recently come from the Cascades and the Colville district in the State of Washington, from the Clearwater country in Idaho, Central and Southern Oregon, and the Lillooet division

of British Columbia. There is reason to believe, also, that the great granite batholith of central Idaho will develop pitchblende when once this mineral is actively sought.

IDENTIFICATION OF RADIUM ORES

The easiest, least expensive and, therefore, the most popular method of identifying radium-bearing ores is by means of the photographic dry plate, as follows:— In a dark-room wrap a sensitive dry plate in two thicknesses of black paper; lay it with the sensitized side up, and on it place any metallic object, such as a flat key; above the metal and just touching it, but not pressing upon it, suspend about three ounces of the suspected ore; enclose the whole in a light-tight box and set aside in a dark place for three or four days. If the ore is appreciably radioactive an image of the metallic object will appear on the plate, in silhouette, when developed.*

"The U. S. Bureau of Mines, 502 Foster Bldg., Denver, will receive any samples giving promise of radium and associated rare metals, and, while it will not undertake chemical analyses for private parties, it will undertake to indicate to them the advisability of further examination and development."

GOVERNMENTAL ACTION RELATIVE TO RADIUM ORES:

Contrary to general belief, the U. S. Government does not contemplate a withdrawal of radium-bearing lands.

Moreover, the bill recommended for passage at the last session of Congress appears, upon cursory examination, to be most favourable to the prospector and miner in that it is calculated not only to encourage prospecting and mining, but also to furnish a ready and steady market for the output;

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* I have found that by working in a dark room by the light of a red lantern, better radiographs may be taken as follows:—

On the sensitized surface of the dry-plate, carefully place the metallic object; over this (or on it) gently place the ore to be tested (so as not to scratch the plate); on the bottom of a light-tight box place a bit of cardboard as large as the plate, and coated with tungstate; on this cardboard lay the plate, loaded as above; cover and set aside in a dark place for at least 48-54 hours; remove from the box and develop—taking great care not to fog the plate by too long eyposure to the red light, or to the *faintest ray* of *white light.*

the element of paternalism, so feared by the public, seems to have been almost wholly eliminated.

The title of this bill is as follows:—(H. R. 12741, reported for passage without amendment):—

"A Bill to provide for and encourage the prospecting, mining and treatment of radium-bearing ores in lands belonging to the U. S., for the purpose of securing an adequate supply of radium for Governmental and other hospitals in the U. S. and for other purposes."

Briefly analyzed; This bill, as reported, withdraws no land from entry nor does it reserve to the U. S. the ownership of radium-bearing deposits on lands still under governmental control. It simply gives to the general Government a *preferential right* (exclusive or not, at the discretion of the Secretary of the Interior) *to purchase radium-bearing ores*, from lands now owned by it, at such prices (to be fixed by the Secretary of the Interior) as will cover the cost of mining operations and encourage the same, under conditions that only require a full and prompt development of the radium deposits. Thus are established two conditions by no means always enjoyed by the miner, namely, a steady market and sure, prompt payment for the product. In this respect, obviously, the bill places radium ores nearly on a par with gold ores, than which there is no better asset known to mining nor to any other industry for that matter.

It is true, as always happens when the question of so-called governmental interference is raised, that there is a difference of opinion on this subject. It is claimed that the general Government has no right to monopolize the output of radium-bearing ores, but should erect its extraction-works and enter the market in open competition with private bidders, of which there are several, notably the Radium Company of America, the Primos Chemical Company, and the Standard Chemical Company, all of which have extraction works in operation in Pennsylvania and New York. Others claim that the general Government, through the Bureau of Mines, discriminates in negotiating with the newly incorporated

National Radium Institute because the latter is a private institution, controlled by individuals, who thus will specially profit.

However, the settlement of all these disputes may safely be left to Congress, trusting to publicity and public opinion for a wise solution of the problem. The bill is a good one on the face of it, provided, always, that the powers it confer are not abused.

The primary reason for this contemplated action on the part of the Government lies in the great value of radium salts in the treatment of disease, especially of certain forms of cancer, rheumatism, et cetera.

The only other reason, at present, is to secure additional supplies of radium salts for scientific use. It has been demonstrated that any attempt to increase soil productivity by the use of radioactive fertilizers, such as carnotite ores or residues, will fail because the cost is prohibitive.

The German States and Austria-Hungary absolutely prohibit the exportation of radium ores, while Australia, British Columbia and other countries have placed restrictions upon its production and sale. The U. S. government is but following the lead of other nations in attempting to keep its radium ores at home.

The phenomena of radiology are closely allied to those of radioactivity, and their application, in the treatment of disease, demands the skill of the physicist as well as that of the physician.

As previously stated, radium emits energy in the form of "rays" that are extremely powerful in their action on organic matter, although wholly invisible to the unaided eye. By means of an instrument called either the "spinthariscope," or "scintilloscope," these emanations may be seen bombarding zinc sulphide screens in a manner that is truly marvelous, resembling nothing so much as a magnificent display of fireworks in which brilliant, uncolored bombs and roman-candles are exploded. These mineral emanations are incessant, and in certain instances they diminish in no appreciable degree for years, and even for ages. Neither do they produce any physical change in the radioactive substances themselves,

so far as the unaided human senses can detect. These radiations from radium, meso-thorium and, perhaps, other combined elements yet undiscovered, are known to scientists under the name of the Alpha, the Betta and the Gamma rays, each of which has its own peculiar action upon organized tissue. It will be recalled, perhaps, that an organic 'cell' is any one of a myriad of minute protoplasmic masses that make up all organized tissues—that life and death, themselves, are but the upbuilding, or the breaking down, of these cells.

Metabolism is that unceasing change in living organisms induced by the action of these cells. Anabolism is constructive metabolism; catabolism is destructive metabolism—in other words, an anabolic process builds up and a catabolic process tears down. Now the Alpha rays of radium are catabolic, that is, they destroy healthy tissue. They are the least penetrating (shortest), of the three and they are checked by paper, glass, aluminum, lead and other metallics. The Beta rays are longer and they are anabolic, stimulating proliferation which is the formation and reproduction of cells. These are checked by lead. The Gamma rays are the longest—they penetrate deepest, are not stopped by lead, and travel at the rate of 185,000 miles per second. Apparently they possess the dual property of checking, and even destroying, *abnormal* growths, while, at the same time, stimulating *normal* growth. These, then, are the rays that are used in the treatment of cancerous growths, after the Alpha rays have been entirely suppressed, and the Beta rays partially so. Medical opinion is not yet in accord in respect to the unqualified value of radium rays in therapy, but the tendency is favourable, and, should the final verdict be unanimous, there would not, perhaps, be sufficient radium in the world to supply the demand for such a specific, and, consequently, spurious remedies would flood the market. In fact, there has already been adduced so much apparently indisputable evidence of the medicinal value of radioactive substances in special cases—not only of radioactive ores, salts and metals, but also of waters as well—that a genuine 'radium rage' is sweeping over the civilized world, and 'radium cures' that never cured anything, nor ever will, are now ready and waiting by the carload

for the coming of the usual crop of gullibles. 'Radium Mud,' for rheumatism, is even now scarce in New York at $3.00 per pound and the chances are that, before long, there may be ample use for a 'Blue Sky' law. Furthermore, in justification of the Radium Bill, the astonishing fact is disclosed that, whereas the U. S. furnishes more than three times as much radium-bearing ore as does all the rest of the world together. nearly every ton of this valuable raw material goes abroad at prices entirely incommensurate with the intrinsic value of its contained radium—and not only that, but if it returns to this country at all (in the form of radium salts), it is only at a price that is nearly prohibitive. Often it is impossible for American surgeons to purchase the salts at any price, so great is the foreign demand for them; in fact, the total quantity at present available for all American hospitals does not exceed two grams—equal to two-fifths the weight of a five-cent piece—whereas the amount needed is estimated at twenty-five grams for hospitals alone. Inasmuch as there are estimated to be more than 200,000 cases of cancer in the U. S., of which 75,000 persons die annually, the importance of these salts and the seriousness of the situation for American patients can scarcely be over estimated.

THE BUSINESS AND MINING PROPOSITION

From the preceding statements, it will be apparent that owing to the comparative rarity of radium-bearing ores, the total output thereof will never approach in value to that of ores of the baser metals; yet the profits possible from radium ores are sufficiently attractive.

Concerning the price of radium, Chas. L. Parsons, in his paper read before the American Mining Congress at Philadelphia, in October, 1913, says: "The price of radium appears to have been holding steadily at about $120 per milligram of metallic radium ($3,401,952 per ounce Avoirdupois), or, in terms of its salts, in which forms it is marketed, at $91,000 per gram for radium chloride ($2,579,813 per oz. Av.), and $70,000 per gram of anhydrous radium bromide ($1,984,472 per oz. Av.), the chloride containing 76% of radium and the bromide, 59%."

In pure pitchblende, which is a primary mineral, the ratio of uranium to radium is constant; in other words, 1 ton of pure uranium oxide will contain 342 milligrams of radium-chloride, and if the percentage of uranium is known by analysis, the amount of radium present may be calculated directly, provided only that the uranium and radium be in equilibrium. Therefore, a fairly typical ore from Paradox valley, containing 3% uranium oxide and 4% of vanadium oxide, would work out about as follows, no allowance being made for losses in treatment:

3% Uranium Oxide—60lb per ton, @ $3.00 . .	$180
4% Vanadium Oxide—80lb per ton, @ $0.75 . .	60
60lb Uranium Oxide with 8.1 mg. ($RaCl_2$) @ $90. per mg. .	729
Total	$969

Compare these figures with the market price of radium ores and it will be seen that the big profit arises from the extraction and refining of the radium salts which are said to cost not more than $20.00 per milligram on carnotite ores. At present, an ore carrying 2% uranium oxide is worth about $2.50 per pound of contained uranium oxide, f.o.b. Hamburg, plus about 13 cents per pound of vanadium oxide content, or about $110 per ton. The latest Denver quotations are $2.50 per pound for 2% ore, $3 for 3%, $3.25 for 20 to 30% and $4.25 for 50 to 70%.

The average cost of laying down a Paradox valley ore at Hamburg is about $70 per ton, freight from Colorado to Hamburg via Galveston, being $14.50, local freight from mines to rails from $18 to $20, and the cost of mining, sorting and sacking, from $28 to $40 per ton. This would indicate a satisfactory profit of some $40 per ton, net, even under all the disadvantages of the present system of mining and marketing. Consequently, as Parsons again remarks: "When it is considered that the Hamburg price for this ore is less than one-sixth of the value of its radium content and that, out of this fraction, the American miner has to meet the outlay represented by investment, mining and assay costs, trans-

portation and loss in transit, it seems scarcely right that over five-sixths of the total value should go to foreign manufacturers of radium salts, especially when it is considered that the latter may be made much more easily from carnotite than from pitchblende. There are two practical ways of reducing this difference between actual value and the price realized by the miner—one being to hold the American ores for higher prices and the other, to manufacture the radium at home." All will agree, it is believed, that the latter way is the better one.

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